A HANDBOOK OF GOLD MILLING

A Stamp Mill in the Sixteenth Century.

(*See page* 94.)

A HANDBOOK

OF

GOLD MILLING

BY

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THIRD EDITION

London
MACMILLAN AND CO., LIMITED
NEW YORK: THE MACMILLAN COMPANY
1902

Richard Clay and Sons, Limited,
london and bungay.

First Edition, 1893
Second Edition, 1899
Third Edition, 1902

PREFACE TO THE FIRST EDITION

THE process of differentiation, so essentially characteristic of all modern development, has of recent years extended very markedly into the province of the gold-miner. The time is not so long past since any ordinary miner was considered to be quite capable of taking a shift in the mill, and it was looked upon as part of his duty to be equally useful above and below ground. This state of things did well enough as long as rich reefs, or rich portions of reefs, such as the superficial portions not uncommonly are, were alone worked; enough gold was won to more than pay expenses, and no one knew or even seemed to care how great the loss might be in the mill. With the rapid exhaustion of high grade reefs and the introduction of more scientific methods of gold extraction, which (together with improved methods and machinery in mining proper) have rendered the exploitation of low-grade ores profitable and therefore possible, a gradual change

has made itself felt within the last few years. At the same time a class of men has gradually come into existence, and has been educated—by the hard lessons of practical experience for the most part—to carry out the duties of amalgamator, reduction officer, or mill man, these being the various names by which this post is designated in various parts of the world—the very divergence in nomenclature serving to show how very recently the need for and existence of such a trade have come to be recognised. Under these circumstances it is hardly to be wondered at that the technical education of the mill man still leaves much to be desired, and the main object of the present volume is to impart so much of such technical instruction in his duties as can be included within the covers of a book. Incidentally it is hoped that it may also prove of service to managers and managing directors of mines, not only by showing them the most general causes of loss both of money and material in gold-milling, and by indicating the readiest means of preventing such loss, but also by pointing out to them what are the essential portions of a mill man's duty and to guide them in selecting a duly qualified man for this onerous post, upon the proper filling of which the prosperity of a mine is so largely dependent. For whilst it is quite true that no one can make a poor mine into a good one, the converse of the proposition is unfortunately by no means impossible; and it is an undoubted fact that there is in existence to-day a large number of mines which would give far

better results than they actually do, were the charge of the mill handed over to a properly qualified mill man. It is not too much to say that not one manager in ten knows how much gold he is losing in his mill—what he is getting is really a secondary question—and this, the fault in the first instance of an imperfectly trained mill man, renders it impossible to detect the shortcomings of the latter, whose very incapacity thus serves as its own shield.

It is only quite recently that the science of gold-milling has come to be at all fully recognised. In most cases this work has been carried on by the light of untrained instinct, the *ignis fatuus* that has so frequently led the way to destruction. It is the object of the writer to apply the results of modern science so as to replace as far as possible the uncertain methods of crude empiricism, and especially to substitute scientific for unscientific modes of reasoning. Facts can usually be trusted to take care of themselves, but the same cannot be said of the methods by which these facts are arrived at. Scientific reasoning is a process far less costly and quite as reliable as unscientific groping in the dark, and the first step towards the foundation of such science is the classification of our knowledge on a systematic basis. This is the main object that has been attempted in the following work, which commences by an account of the physical and chemical properties of gold and also of mercury, the knowledge of the latter being of scarcely less importance than that of the former

for the scientific amalgamator; stamp mill construction is considered in detail, the mechanical principles underlying the design of each part being throughout elucidated. The theory and practice of concentration, as far as it refers to gold-milling, is next considered, together with the most approved modern method of treating the concentrates and the other products of milling. A chapter on the economic considerations involved, and one on the assaying of gold ores and mill products are also appended.

The art of gold-milling rests upon a twofold scientific basis, the comminution of the ore being a purely mechanical and the extraction of the gold a purely chemical process. The mill man should accordingly have a knowledge of the theories of both Mechanics (including Physics) and Chemistry. Practically he should have served his time as an engineer, or at least have gone through the shops sufficiently to be a good fitter and a fair carpenter; he should also have had some experience in looking after machinery in motion. In addition to the above, he ought to have worked for a year or so in a chemist's or assayer's laboratory, where he will have acquired habits of accuracy in chemical manipulation, and where he ought to have learnt some assaying. He is then duly prepared to undertake the duties of a mill man, and the writer hopes that a study of this volume will teach him exactly what are the problems which he will be called upon to solve, and what are at present the most

approved methods of solving them. Much of the information here given may be found scattered through various British, American, and foreign works on Metallurgy, and more especially in the Transactions of Societies devoted to the cultivation of this and cognate sciences. So very many of the facts contained in this volume have become the common property, so to speak, of metallurgists, that, having regard, moreover, to the essentially practical purposes of the present work, it has been considered advisable not to encumber its pages with references to all the sources whence information has been derived; the author accordingly hopes that this general acknowledgment of obligation to previous writers on this subject will be deemed sufficient.

HENRY LOUIS.

LONDON,
November, 1893.

PREFACE TO THE SECOND AND THIRD EDITIONS

As the general arrangement of this work seems to have answered its purpose fairly well, I have left it as far as possible untouched, merely bringing it thoroughly up to date. The greatest difficulty has been experienced in keeping down the bulk of the book on account of the great advances that have been made in the last few years in the art of gold extraction; it is at the same time gratifying to find that many of the recommendations made in the first edition have now passed into current gold-milling practice.

HENRY LOUIS.

NEWCASTLE-UPON-TYNE,
June, 1899.
August, 1902.

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GOLD MILLING

CHAPTER I

MODE OF OCCURRENCE—DEPOSITS—PARAGENESIS—VARIOUS FORMS OF GOLD

GOLD is a metal that has been known to, and prized by, mankind from pre-historic times. Occurring as it does in the native state, its great specific gravity and well-marked colour and lustre must have attracted the attention of savages, even at a very early stage of the development of their powers of observation.

At a period when the extraction of other metals from their ores would have proved a task impossible to their rudimentary resources, the great softness, malleability, and ready fusibility of gold (the latter probably discovered after brief experience) would have combined to render it of high value to them for many purposes. And yet, in spite of its having been known and worked for so long a period, the scientific metallurgy of gold is in reality but little more than half a century old. Up till then there seems to have been little or no improvement in the methods employed for its extraction since mediæval times, gold working and especially gold milling having

remained rude and neglected industries ntil the great discoveries of gold in California in 1849, and almost immediately afterwards those in Australia, turned the energies of thousands of able, active, intelligent men into this new channel. In the mad race for wealth which ensued, the primitive methods, all sufficient up to that time, were soon found to be too crude and too slow to satisfy the ambitions of modern gold diggers, whose inventive faculties, stimulated to the highest pitch, were prolific in new methods and improvements upon the old ones. When the rich shallow placers and the rich superficial pockets of gold quartz had been exhausted, and it became necessary to work the poorer alluvials and low-grade reefs, the utmost resources of modern science became even more indispensable, and the present complex system of gold extraction is the ultimate result. Of course this industry is even yet in a comparatively rudimentary and backward state; the stamp-mill is at best a somewhat cumbersome machine, and the science of the subject is even now but imperfectly understood. Nevertheless the art is very far from stationary, and steady progress is being made in various directions, with the result that sources of gold are now being profitably worked which are actually poorer than the tailings, which were a few years ago allowed to run to waste as worthless.

Occurrence of Gold.—The manner in which gold occurs in nature may be divided for the purpose of the present work into two classes, this division being neither geological nor mineralogical, but simply technical, dependent upon the methods employed for extracting the precious metal from the matrix that carries it:

Class A.—Gold occurring in a matrix capable of complete disintegration by the action of a stream of water.

Class B.—Gold occurring in a firm matrix, which requires the employment of mechanical appliances for its pulverisation.

Class A.—In this is included the occurrence of gold in alluvial deposits of various ages, in river sands, on sea-shores, and in gravels of various characters; the gold, which occurs in extremely small proportions as a general rule, is readily extracted by the comparatively primitive method of washing. When the mass, consisting of gold in admixture with the lighter substances with which it is associated, is agitated in a stream of water, the relatively heavy particles of gold sink to the bottom, whilst the relatively light sand, pebbles and clay are carried off by the current. It is by this method, or some variety thereof, that gold has always been obtained by primitive races of mankind. The washing was originally, no doubt, performed in a dish of some kind, and subsequently in a trough of greater or less length. In time the former developed into the cradle, which consists of a short rectangular box to which a sieve is fitted, and which is supported upon rockers, thus receiving a lateral undulating motion, whilst the auriferous gravel, together with water, is thrown into it. The latter method developed into the ground sluice, which is simply a channel cut into the ground itself, and is often of very great length; and finally American ingenuity produced the various methods of "hydraulicking," which consists in projecting a powerful stream of water, brought in pipes from a considerable elevation, against the face of a gravel deposit, and then washing down the latter at a very rapid rate, and at a minimum of cost. Mercury is frequently used in most of the modern methods to assist in the separation of gold from the other materials with which

it is associated. All the above methods are usually included under the general head of "Hydraulic Gold Mining." Their consideration is entirely foreign to the purposes of the present work, which is concerned exclusively with the methods employed for the treatment of such auriferous deposits as are included in Class *B*.

Class B.—This class comprises the following modes of occurrence, which differ, as will be seen, more in the manner of their origin than in their intrinsic composition; in other words, this class is subdivided on geological rather than on mineralogical principles.

(1.) *Reefs* (known also as lodes, leads or veins).—These may be of any of the well-known types of mineral veins, fissure veins being perhaps the most abundant (Australia, Nova Scotia), though contact veins (Montana) and intercalated veins (Nova Scotia) also frequently occur; a variety of this form of occurrence is that in which a stratified deposit, highly metamorphosed, is traversed by a reticulated mass of veinlets which carry gold. As a notable example of this form, the well-known "Great Mother Lode" of California may be quoted. The country rock is mostly slate or schist; sometimes the auriferous veinlets are carefully picked out and treated separately, but more often the entire mass of rock and reef is sent to the mill, the method adopted varying with the richness, size, and physical conditions of the deposit. The principal portion of the gangue of which auriferous veins are composed, is always quartz.

(2.) *Metamorphosed Ancient Stratified Deposits.*—These are at times only an extreme variety of the occurrence last mentioned, where schists or shales, either in the neighbourhood of quartz reefs, or of which such veins form a larger or smaller portion, are impregnated with

gold. Again, deposits which have by extreme metamorphism passed into quartzites, sometimes carry gold; a notable example of this is the so-called "Sheba reef," near Barberton, South Africa.

(3.) *Gravel Deposits of Various Ages*, such as are included under Class *A*, are sometimes found consolidated by metamorphic action, or cemented by various binding materials into an auriferous conglomerate, occasionally very hard. In these the gold may be either contemporaneous with the pebbles forming the deposit, and is then like them rounded and waterworn, as is the case in the "Cement Mines" of California, or else it has been introduced subsequently to the deposition of the bed, when it presents a crystalline structure, as in the well-known "Banket" of Witwatersrand, South Africa.

(4.) *Eruptive Rocks.*—Some deposits of auriferous eruptive rocks are known, and worked; these rocks are generally acidic, like the granites of Treadwell, Alaska, U.S.A.; Timbarra, New South Wales; Beresof, Urals; Sonora, Mexico; Cripple Creek, Colorado, &c.; in some at any rate of these instances it would appear that the gold may have formed a primary constituent of the rock mass. Gold has also been said to occur in basic eruptives like diorite and basalt, but the evidence of these occurrences is less convincing.

Whatever their origin, all these divisions of Class *B* consist essentially of a siliceous matrix carrying gold, mostly in admixture with small proportions of other minerals. Quartz is the universal vein-stuff of gold, and as practically all other modes of occurrence of gold appear to be derived from the disintegration of such veins in the first instance, clay being mostly present in some form, the chemical composition of all the above divisions is prac-

tically identical, being mostly silica, together with more or less silicate of alumina, oxide of iron, &c. Among gold miners the practice mostly obtains of speaking of an auriferous matrix, whatever be its real nature, as "quartz." The term is of course inexact, but at the same time it is quite true that the principal portion of such matrix does really consist of silica; it is also worth noting that in general the gold miner indiscriminately calls all these various occurrences "reefs," whether they be true veins or not.

The following table gives an approximate estimate of the amount of gold produced in the world from deposits of Class *A* and Class *B* respectively in the year 1897:

	Class *A*. Alluvial Gold.		Class *B*. Reef Gold.	
	Fine Gold. Ounces.	Value in Sterling.	Fine Gold. Ounces.	Value in Sterling.
Australasia	418,000	£1,775,500	2,170,000	£9,217,000
Africa	35,000	148,500	2,765,000	11,744,500
United States	470,000	1,996,500	2,400,000	10,194,000
Rest of North America	330,000	1,401,000	340,000	1,444,000
South America	418,000	1,775,500	180,000	764,500
Europe (incl. Siberia)	876,000	3,721,000	295,000	1,253,000
Asia	380,000	1,614,000	410,000	1,741,000
Totals	2,927,000	£12,432,000	8,560,000	£36,358,000

Minerals Associated with Gold.—There are a large number of minerals that accompany gold in the deposits of Class *B*, some of which are frequently, and some but rarely, associated with gold. It is important that these minerals should be thoroughly known, and their properties carefully studied, as their presence or absence may exert a very marked influence upon the milling of the ore.

These associated minerals may conveniently be divided into four groups:

A. Non-metallic minerals.
B. Metallic minerals, rarely or never auriferous.
C. Metallic minerals, frequently auriferous.
D. Minerals containing gold as an essential element of their composition.

The minerals of Classes *B* and *C* occur with gold, and frequently show visible metallic gold enclosed in their substance; those of Class *B* rarely contain gold in a state in which it is held invisibly, whereas Class *C* often holds gold in this condition. The following is a list, as complete as possible, of all minerals that have been known to occur in gold-bearing veins. The rare minerals, which are only known in a few special localities and need not be looked for as a general rule, are in small type, the common ones being distinguished by full-faced type. Many of these minerals are, it will be noted, decomposition products of others that are more essentially the true original constituents of auriferous reefs, and their occurrence is accordingly limited to such portions of the deposits as lie above water level and have been subjected to atmospheric influences. Thus in Class *A*, sulphur is a decomposition product of iron pyrites, and selenite may result from the oxidation of the same mineral, the sulphuric acid so formed in its turn decomposing calcite. A very large number of the minerals comprised in Class *B* are the results of decomposition or oxidation of some of those in Class *C*. Whenever these secondary minerals are found in the upper portion of a deposit, the minerals from which they are derived may be expected to occur in depth.

A.—NON-METALLIC MINERALS.

Name.	Chemical Composition.	Crystallisation.	Hardness.	Specific Gravity.	Localities.	—
Quartz	SiO_2	**Rhombohedral**	**7.**	**2·6 to 2·65.**	...	The universal constituent of gold reefs.
Clays	**Hydrated Silicate of Alumina**	...	...	**2·6 to 2·63.**		
Chlorite	$\mathbf{4H_2O, 5(MgFe)O, Al_2O_3, (SiO_2)_3}$	**Monoclinic**	**2 to 2·5.**	**2·6 to 2·85.**		
Mica (Muscovite)	$K_2O, 2H_2O, 3Al_2O_3(SiO_2)_6$	Monoclinic	2 to 2·5.	2·76 to 3.		
Talc (Steatite)	$H_2O, 3MgO, (SiO_2)_4$	Rhombic or Monoclinic.	1 to 1·5.	2·7 to 2·8.		
Serpentine	$3MgO, 2H_2O, (SiO_2)_2$	Massive	2·5 to 4.	2·3 to 2·65.	...	Occasionally forms the casing or walls of auriferous quartz veins.
Felspar (Orthoclase)	$\mathbf{K_2O, Al_2O_3(SiO_2)_6}$	**Monoclinic**	**6.**	**2·6.**	**Colorado.**	
Garnet	$3FeO, Al_2O_3(SiO_2)_3$	Isometric	6·5 to 7·5.	3·9 to 4·2.		
Grossularia	$3CaO, Al_2O_3(SiO_2)_3$	Isometric	6·5 to 7·5.	3·5 to 3.65.	Siam.	
Calcite	$\mathbf{CaCO_3}$	**Rhombohedral**	**3.**	**2·71 to 2·72.**		
Dolomite	$(CaMg)CO_3$	Rhombohedral	3·5 to 4.	2·8 to 2·9.	California, Transylvania.	
Aragonite	$CaCO_3$	Rhombic	3·5 to 4.	2·93 to 2·99.	New Zealand.	
Magnesite	$MgCO_3$	Rhombohedral	3·5 to 4.	3·0 to 3·2.	Victoria.	
Apatite	$3(Ca_3P_2O_8)+CaF_2$	Hexagonal	5.	3·17 to 3·23.	Ottawa.	
Tourmaline	Borosilicate of alumina, magnesia, iron, &c.	Rhombohedral	7 to 7·5.	2·98 to 3·2.		
Hornblende	Silicate of magnesia, lime & iron.	Monoclinic	5 to 6.	2·9 to 3·4.		
Fluorite	CaF_2	Isometric	4.	3·01 to 3·25.		
Barytes	$BaSO_4$	Rhombic	2·5 to 3·5.	4·3 to 4·6.		
Prehnite	$H_2O, 2CaO, Al_2O_3, (SiO_2)_3$	Rhombic	6 to 6·5.	2·8 to 2·95.	Craddock, Cape Colony.	
Pyrophyllite	$H_2O, Al_2O_3, (SiO_2)_4$	Monoclinic (?)	1 to 2.	2·8 to 2·9.	Beresovsk.	
Sulphur	S	Rhombic	1·5 to 2·5.	2·05 to 2·09.	,,	
Selenite	$CaSO_4, (H_2O)_2$	Monoclinic	1·5 to 2.	2·31 to 2·33.		
Lignite	Carbon, hydrocarbons, &c., &c.	Massive	0·5 to 1·5.	1·0 to 1·5.	Transylvania.	

Name.	Chemical Composition.	Crystallisation.	Hardness.	Specific Gravity.	Localities.	—
Ilmenite	**$FeTiO_3$**	**Rhombohedral**	**5 to 6.**	**4·5 to 5.**		
Magnetite	**Fe_3O_4**	**Isometric**	**5·5 to 6·5.**	**5·17 to 5·18.**		
Haematite	Fe_2O_3	Rhombohedral	5·5 to 6·5.	4·2 to 5·3.		These are decomposition products of, and at times pseudomorphous after, Pyrites.
Limonite	**$2Fe_2O_3, 3H_2O$**	**Massive, earthy, &c.**	**5 to 5·5.**	**3·6 to 4.**	...	
Siderite	$FeCO_3$	Rhombohedral	3·5 to 4.	3·83 to 3·88.		
Vivianite	$Fe_3P_2O_8, 8H_2O$	Monoclinic	1·5 to 2	2·58 to 2·68.	Victoria.	
Manganese blende	MnS	Isometric	3·5 to 4.	3·95 to 4·04.	Transylvania.	
Rhodocroisite	$MnCO_3$	Rhombohedral	3·5 to 4·5.	3·45 to 3·65.	Transylvania.	
Bustamite	$(MnCa)SiO_3$	Triclinic	5·5 to 6·5.	3·4 to 3·6.	Mexico.	
Pyrolusite	MnO_2	Rhombic	2 to 2·5.	4·73 to 4·86.		
Chrysocolla	$CuSiO_3, 2H_2O$	Cryptocrystalline	2 to 4.	2 to 2·24.	Namaqualand	These are decomposition products of sulphuretted copper ores.
Malachite	$CuCO_3, CuH_2O_2$	Monoclinic	3·5 to 4.	3·9 to 4·03.	Peru, &c.	
Azurite	$2CuCO_3, CuH_2O_2$.	Monoclinic	3·5 to 4.	3·77 to 3·83.	Peru, &c.	
Wolfram	$(MnFe)WO_4$	Monoclinic	5 to 5·5.	7·2 to 7·5.	Victoria.	
Scheelite	$CaWO_4$	Tetragonal	4·5 to 5.	5·9 to 6·1.	Italy, Bohemia, &c.	
Pyromorphite	$3(Pb_3P_2O_8) PbCl_2$	Hexagonal	3·5 to 4.	6·5 to 7·1.	Wales.	
Mimetite	$3(Pb_3As_2O_8)PbCl_2$	Hexagonal	3·5.	7 to 7·25.	Nevada.	
Crocoisite	$PbCrO_4$	Monoclinic	2·5 to 3.	5·9 to 6·1.	Peru.	
Cervantite	Sb_2O_4	Rhombic	4 to 5.	4·08.	Victoria.	These are decomposition products of Stibnite.
Valentinite	Sb_2O_3	Rhombic	2·5 to 3.	5·56.	Victoria.	......
Kermesite	$2Sb_2S_3, Sb_2O_3$	Monoclinic	1 to 1·5.	4·5 to 4·6.	Victoria.	
Bismuthite	$Bi_2O_3CO_2, H_2O$	Amorphous, earthy	4 to 4·5.	6·86 to 6·9.	Transvaal.	
Native Lead	Pb	Isometric	1·5.	11·37.	Victoria.	
Native Bismuth	Bi	Rhombohedral	2 to 2·5.	9·7 to 9·83.		
Native Arsenic	As	Rhombohedral	3·5.	5·63 to 5·73.		
Native Silver	Ag	Isometric	2·5 to 3.	10·5.	Mexico.	
Aikinite	$PbCuBiS_3$	Rhombic	2 to 2·5.	6·1 to 6·8.	Beresovsk.	
Melanochroite	$3PbO2CrO_3$	Rhombic (?)	3 to 3·5.	5·75.	,,	
Vauquelinite	$2(PbCu)Cr_2O_4(PbCu)_3P_2O_8$	Monoclinic	2·5 to 3.	5·8 to 6·1.	,,	
Vanadinite	$3Pb_3V_2O_8, PbCl_2$	Hexagonal	2·75 to 3.	6·66 to 7·23.	,,	
Cerussite	$PbCO_3$	Rhombic	3 to 3·5.	6·46 to 6·57.	,,	
Anglesite	$PbSO_4$	Rhombic	2·75 to 3.	6·12 to 6·39.	,,	
Scorodite	$Fe_2O_3, As_2O_5, 4H_2O$	Rhombic	3·5 to 4.	3·1 to 3·3.	,,	
Jarosite	$K_2O, 3Fe_2O_3 (SO_3)_4, 6H_2O$	Rhombohedral	2·5 to 3·5.	3·15 to 3·26.	,,	

C.—AURIFEROUS METALLIC MINERALS.

Name.	Chemical Composition.	Crystallisation.	Hardness.	Specific Gravity.	Localities.
Iron Pyrites	**FeS_2**	**Isometric**	**6 to 6·5.**	**4·95 to 5·1.**	
Mispickel	**FeS_2, $FeAs_2$**	**Rhombic**	**5·5 to 6.**	**5·9 to 6·3.**	
Löllingite	$FeAs_2$	Rhombic	5 to 5·5.	6·8 to 7·4.	Idaho.
Copper Pyrites	$CuFeS_2$	Tetragonal	3·5 to 4.	4·1 to 4·3.	
Erubescite	Cu_3FeS_3	Isometric	3.	4·9 to 5·4.	Russia.
Covelline	CuS	Rhombohedral	1·5 to 2.	4·59 to 4·64.	
Redruthite	Cu_2S	Rhombic	2·5 to 3.	5·5 to 5·8.	
Stibnite	**Sb_2S_3**	**Rhombic**	**2.**	**4·52 to 4·62.**	
Galena	**PbS**	**Isometric**	**2·5 to 3.**	**7·4 to 7·6.**	
Zinc Blende	**ZnS**	**Isometric**	**3·5 to 4.**	**3·9 to 4·1.**	
Cinnabar	HgS	Rhombohedral	2 to 2·5.	8 to 8·2.	
Bournonite	$PbCuSbS_3$	Rhombic	2·5 to 3.	5·7 to 5·9.	
Tetrahedrite	$Cu_3Sb_2S_7$	Isometric	3 to 4·5.	4·4 to 5·1.	California, &c.
Pyrargyrite	Ag_3SbS_3	Rhombohedral	2·5.	5·77 to 5·86.	
Molybdenite	MoS_2	Hexagonal	1 to 1·5.	4·7 to 4·8.	
Tetradymite	$Bi_2(TeS)_3$	Rhombohedral	1·5 to 2.	7·2 to 7·6.	Hungary, Norway.
Argentite	Ag_2S	Isometric	2 to 2·5.	7·2 to 7·36.	Siberia.
Clausthalite	$PbSe$	Isometric	2·5 to 3.	7·6 to 8·8.	Harz.
Hessite	Ag_2Te	Isometric	2·5 to 3.	8·31 to 8·89.	Transylvania.
Altaite	$PbTe$	Isometric	3 to 3·5.	8·1 to 8·2.	Burma.
Melonite	Ni_2Te_3	Hexagonal	...	...	California.
Stromeyerite	Ag_2SCu_2S	Rhombic	2·5 to 3	6·15 to 6·3.	Siberia.
Native Tellurium	Te, with a little gold and iron	Rhombohedral	2 to 2·5.	6·1 to 6·3.	Transylvania.

D.—MINERALS CONTAINING GOLD AS AN ESSENTIAL CONSTITUENT.

Name.	Chemical Composition.	Crystallisation.	Hardness.	Specific Gravity.	Localities.
Petzite	$(AgAu)_2Te$	Massive	2·5 to 3.	8·7 to 9·02.	Transylvania, Colorado, California.
Krennerite	Telluride of gold and silver	Rhombic	2·5.	8·35.	Transylvania.
Calaverite	,, ,,	Massive	2·5.	9·04.	California.
Sylvanite	$(AgAu)Te_2$	Monoclinic	1·5 to 2.	7·9 to 8·3.	Transylvania, California, Colorado.
Nagyagite	$Au_2Pb_{14}Sb_3Te_7S_{17}$	Rhombic	1 to 1·5.	6·85 to 7·2.	Transylvania.
Kalgoorlite	$HgAu_2Ag_6Te_6$	....	....	8·79.	Kalgoorlie (W. Australia).

Of the physical characters of the above minerals, those of chief importance for the mill-man to bear in mind are their hardness and specific gravity, as these influence, the former the crushing, and the latter the concentrating of the ore that carries them.

The hardness of minerals is their power of resisting abrasion; thus a harder will always scratch a softer mineral, and if two pieces of mineral are rubbed together, the softer will lose more weight than the harder one. Hardness is indicated in the above list by Mohs's scale of hardness, well known to all mineralogists. It need only be said in this place that the numbers are quite empirical and have no quantitative value. They merely indicate the relative order, but not the proportion of hardness. A hardness of 3 on the scale is about that of a plate of copper, 6 is about that of ordinary glass, 7 is that of the hardest steel; iron varies between 5 and 7. The hardness of quartz, the principal constituent of all gold-bearing reefs, being thus greater than that of the materials from which the crushing machinery for the same is constructed, it follows that the wear of the latter must always be very considerable, and that this is unavoidable from the very constitution of the substances themselves.

Disregarding the minerals which but rarely accompany gold, and turning our attention solely to those that are usually associated with it, it will be noticed that the average specific gravity of Class *A* is under 2·7; of Class *B* it is about 4·5 to 5, and of Class *C* it is over 5; an admixture of the telluride minerals that contain gold comprised in Class *D*, would raise this figure still higher. The importance of this matter will be better appreciated further on; for the present, it will be sufficient to note

that as a general rule minerals with which gold is intimately associated (Classes *C* and *D*), have a specific gravity of about 5. These minerals are usually the object of a special series of operations for their collection with a view to the extraction of the gold they contain. When thus collected they form the "concentrates" or "sulphurets" of the gold miner; either term may be and is used indifferently, but the former is certainly the more correct. These concentrates may obviously contain any of the minerals of Classes *C* and *D*, and they will also at times contain some of the heavier ones of Class *B*, notably magnetite; these latter minerals must, however, be looked upon as diluents or impurities of the concentrates proper. It is interesting to note that marcasite seems to occur only quite exceptionally in auriferous veins, although pyrites is such a very common constituent.

The minerals of Class *D* were until quite recently looked upon as rarities; they have now, however, been discovered in considerable quantities in the Cripple Creek region of Colorado, and in various of the gold fields (notably Kalgoorlie) of West Australia, in both of which regions the so-called "telluride ores" are producing considerable quantities of gold. These telluride ores may contain any or all the minerals of Class *C* containing tellurium, together with more or less of those of Class *D*, and perhaps also some other ill-defined mineral species, the exact composition of which has not yet been determined.

Mode of Occurrence of Reef Gold.—Native gold occurs in reefs in particles of all sizes, from masses of over 100 lbs. in weight (Meroo Creek, New South Wales) down to specks that are only visible under powerful magnifica-

tion. It but seldom occurs well crystallised; it crystallises in the isometric system, the commonest forms being the octahedron, cube, and dodecahedron. The crystals are rarely distinct, mostly aggregated, and frequently distorted and elongated in the direction of one of the axes. It more often occurs in irregular grains, plates, scales, filiform, dendritic, reticulated, or spongy. Sometimes it is pseudomorphous after some other mineral, mostly iron pyrites.

It has no cleavage, and a hackly fracture.

In nature gold never occurs in an absolutely pure state; it is either alloyed with various metals or else, more rarely, is combined with a few metalloids in the scarce minerals of Class *D*. The principal metals with which it is alloyed are silver, copper, and iron, the former being universally present.

Native geld thus consists of an alloy of gold with silver or copper, or both, the name of gold being applied to such alloys as long as the gold is present in such quantity that its value considerably exceeds that of the other constituents. No hard and fast line of division between *e.g.* argentiferous gold and auriferous silver has yet been drawn, the distinction being essentially a technical one, depending upon the methods by which the ore has to be treated (whether by gold- or by silver-extraction processes) and upon the character and after treatment of the bullions produced. I would suggest that a useful practical line of demarcation could be drawn at the alloy containing forty-eight parts of gold to fifty-two of silver, which is of just about the same specific gravity as mercury. Argentiferous gold with more gold than the above would therefore sink in mercury, whilst auriferous silver, with less gold than the above, would

float on it. The alloy containing four parts of gold to one of silver is called electrum. Silver with 10 to 30 per cent. of gold is called küstelite.

The specific gravity, hardness, colour, and other physical characters of the alloy depend, of course, upon its composition. Some ambiguity may occasionally result from this, strictly speaking, incorrect use of the word gold, to designate both the pure and the native impure metal; but it has been sanctioned by custom, and the context will mostly serve to show whether pure gold or native gold is meant; wherever there might be any doubt, I shall use the expression pure gold to distinguish it from the native alloy.

The table on the next page gives the compositions and specific gravities of a number of different specimens of native gold from various sources; these data have been collected from various publications and are the work of trustworthy observers. They form, of course, only a very small proportion of the total number of analyses of native gold that have been placed on record. Some of the specimens consisted of alluvial nuggets, but most of them were reef gold.

In the following table the analyses of gold have been arranged in descending order, according to the amount of pure gold which they contain. It would naturally be expected that their specific gravities should follow the same order; they certainly do so approximately, but by no means exactly, there being numerous abnormal results, that show specific gravities either markedly higher or markedly lower than the composition of the metal would lead one to expect. The reason for these anomalies is at present quite obscure; all, however, can scarcely be due to errors of observation. They may be caused by unde-

Locality.	Specific Gravity.	Percentage Composition.					Analyst.
		Gold.	Silver.	Copper.	Iron.	Silica.	
Urals	19·10	98·96	0·16	0·35	0·05	...	Rose.
Katharinenburg	18·79	95·81	3·58	0·61		...	Avdejef.
Katharinenburg	15·60	95·48	3·59	...	...	...	Kerl.
Bolivia	18·31	94·73	5·23	...	0·04	...	Forbes.
Bolivia	17·906	93·51	6·49	...	...	...	Forbes.
Sacramento ...	16·23	93·0	6·7	...	...	...	Rivot.
Wicklow	16·342	92·32	6·17	...	0·78	...	Mallet.
Bolivia	16·07	91·96	7·47	...	trace	0·57	Forbes.
Vancouver Island	18·5	91·86	6·63	1·00	...	...	Wibel.
Boruschka ...	17·955	91·36	8·35	0·29		...	Rose.
California... ...	17·40	90·96	9·04	...	...	...	Oswald.
Peru	16·693	90·86	9·14	...	...	...	Forbes.
Boruschka ...	17·588	90·76	9·02	trace	trace	...	Rose.
Clogau, Wales...	17·26	90·16	9·26	trace	trace	0·32	Forbes.
Cornwall	16·52	90·12	9·05	...	...	0·83	Forbes.
Ashantee	17·55	90·055	9·94	trace	trace	...	Church.
California... ...	15·96	90·01	9·01	0·86		...	Henry.
Wales	15·62	89·93	9·24	...	...	0·74	Forbes.
West Africa ...	14·63	89·40	10·07	0·53	...	...	Wibel.
Canada	16·57	89·24	10·76	...	...	...	Kerl.
Colombia	14·69	87·94	12·06	...	...	...	Boussingault.
West Africa ...	16·20	87·91	11·40	0·69	..	...	Wibel.
Canada	17·60	87·77	12·23	...	...	...	Hunt.
Scotland	16·50	86·60	12·39	...	0·35	...	Church.
Boruschka ...	17·061	83·85	16·15	...	...	...	Rose.
Sutherland ...	15·799	81·27	18·47	...	...	0·36	Forbes.
Peru	16·63	79·89	20·11	...	...	...	Forbes.
Katharinenburg	16·03	79·69	19·47	0·84		...	Avdejef.
Scotland	16·62	79·22	20·78	...	...	...	Church.
Colombia	16·54	78·0	20·11	...	...	...	Forbes.
Colombia	12·666	73·45	26·48	...	...	...	Boussingault.
Katharinenburg	15·627	70·86	28·30	0·84		...	Avdejef.
Virginia	15·46	65·31	34·01	0·14	0·20	0·34	Porcher.
Colombia	14·149	64·93	35·07	...	...	...	Boussingault.
California... ...	15·15	63·34	36·41	...	...	...	Hanks.
Urals	14·556	60·98	38·38	...	0·33	...	Rose.

tected cavities or inclusions in the mass of the metal, which would of course reduce the density, or possibly by allotropism of one or other of the constituents.

The purest native gold known is that from Mount Morgan, Queensland, said to contain 99·7 to 99·8 per cent. of gold, a little copper and iron, and a mere trace of silver. An analysis by Leibius gave Au 99·70 per cent., Cu 0·29 per cent. Silver is always present in all native gold, and its amount may be judged from the colour of the metal, unless this is masked by other causes. In the latter case a good comparative test consists in fusing a small particle of the metal in a bead of microscopic salt; a small quantity of silver will make this bead opalescent and yellowish, a larger quantity opaque and distinctly yellow. Less than 0·25 per cent. of silver produces no effect at all.

The average richness of Australian gold is 90 to 92·5 per cent., of Californian gold 88 per cent., of Ural reef gold 92 per cent., of Ural alluvial gold 91 per cent. of pure gold.

Rare Alloys.—In addition to the above-named alloying metals, the following rare native alloys of gold are also known:—

Palladic Gold from Brazil, containing 10 to 25 per cent. of palladium.

Porpezite from Colombia, containing 35 to 60 per cent. of rhodium.

Maldonite from Maldon, Victoria, containing gold 64·5, bismuth 35·5 per cent. (Au_2Bi).

Native Amalgam occurs at Mariposa, California, in small yellow crystals containing: gold 39·02 to 41·63, mercury 60·98 to 58·37 per cent. At Choco in Colombia it also contains silver: gold 38·39, silver 5·00, mercury 57·40, corresponding to the formula $(\frac{4}{5}Au\frac{1}{5}Ag)_4Hg_3$.

Platinum, osmium, iridium, and antimony also occur alloyed with gold.

Modes of Occurrence of Gold.—Besides the two well-marked forms of free native gold and chemically combined gold forming a constituent of the gold-bearing minerals of Class *D*, gold also occurs in other modes of combination with the minerals of Class *C*, the precise nature of which is as yet by no means fully understood. The native sulphides of many metals, such as copper, lead, zinc, &c., at times carry gold, whilst iron and arsenical pyrites are very frequently auriferous. The gold is sometimes visible under the microscope in fine threads or plates penetrating the substance of these minerals, but sometimes occurs in some other form indistinguishable even microscopically. It is by no means clear whether it then exists in these sulphides, chemically combined as a sulphide, or whether it is present in an extremely fine state of mechanical subdivision. On the one hand, microscopic examination will not always reveal its presence, and it cannot be completely extracted by substances which have great affinity for uncombined gold, unless the sulphides be first destroyed by calcination. On the other hand, such substances (mercury for instance) will extract a certain proportion of gold from auriferous sulphides provided these are very finely ground, and will extract a greater proportion the more finely they are subdivided and the longer continued the grinding. Many theories have been put forward as to the mode of existence of this gold, the principal ones being that it is gold in a very fine state of division, more or less encrusted with a film of some substance such as quartz, or the sulphide of some base metal, &c., or that it is in combination with sulphur, forming a complex polysulphide with the other metal or metals present. Such complex sulphides are, of course, known to exist, and the more

stable compound always seems to exert a protective influence upon the less stable, and to prevent the decomposition of the latter under conditions in which it could not by itself exist. Neither of these theories fully explain all the phenomena, and it would seem that the true explanation has yet to be found. The facts are, at any rate, well known: if a gold ore containing quartz, auriferous pyrites, and free gold, be ground up with mercury, this latter will extract a portion of the free gold, whilst another portion will not be attacked. If all the free gold be removed and the pyrites ground up with mercury, some gold will be extracted from the pyrites, and the longer the grinding is continued, the more gold will be obtained, but even the most prolonged grinding will fail to extract the whole of the gold. If, however, the pyrites be calcined until the whole of the sulphur has been driven off, the gold can be readily and completely extracted by means of mercury.

In addition to the above modes of occurrence, there is yet another, namely, of free nonamalgamable gold; this mostly goes by the name of "rusty" gold. It is sometimes metallic-looking, of the usual golden colour, but more often brown and lustreless. Many gold ores carry more or less of their valuable contents in this form, in which it resists the action of mercury, and heavy loss is apt to occur in the treatment of such ores by the usual processes. I have found this rustiness particularly marked in ores in which the gold had been deposited upon decomposing crystals of pyrites, or in cavities left by their complete destruction, producing pseudomorphs of gold after pyrites. No satisfactory explanation of the cause of rustiness has as yet been advanced. It is

generally said to be due to the gold being coated by a film of something or other, variously supposed to be silica, oxide of iron, sulphur, some metallic sulphide, some compound of gold, &c., and the fact that rusty gold will partially (but never by any means completely) amalgamate when exposed to prolonged friction with mercury is explained by saying that the coating has been removed by the attrition.

If, however, the ore be calcined at a red heat, the gold is no longer rusty, but is readily amalgamable, and it is difficult to see how this could be the case were the sole cause of rustiness to be due to a coating of silica or oxide of iron.

Lazarus Ercker, who wrote as far back as 1672, points out that it is beneficial to calcine gold ores before treating them by crushing, &c., because "the fine subtle gold shrinks and runs together, and assumes a rounded *corpus* and such strength that it remains firm in washing an can be caught."

The practice of roasting ores before crushing used at one time to obtain in California, when such ores were being treated in the arrastra. It is, of course, obvious that this mode of procedure is only applicable when the ore is rich and fuel plentiful. If the calcining of the ore costs more than the value of the gold saved by it, the process, however correct it may be scientifically, is practically worthless. We are not, however, now considering the question in its economic aspect, but purely from a scientific standpoint.

It may also be noticed that calcination would be effective if the coating were to consist of a thin layer of sulphide of gold, as is suggested by Skey.[1] This chemist

[1] *Chem. News*, xx. 282.

has studied this question in much detail, and has made numerous experiments bearing on it; he summarises his investigations as follows:—"(1) Samples of clean-looking gold from some reefs refused to amalgamate though taken direct from the reef and untouched by hand. (2) On such surfaces sulphur is always present. (3) Native gold or pure gold readily absorbs sulphur from moist sulphuretted hydrogen or ammonic sulphide, and absorbs it directly when administered in boiling water. (4) Surfaces so treated refuse to amalgamate, though no apparent change is perceptible in their aspect. (5) Gold so affected is rendered amalgamable by heating in an open fire (unless it contains over 7 per cent. of copper); the same effect is produced by the contact of potassic cyanide, chromic and nitric acid, and chloride of lime acidified. (6) This absorption is altogether of a chemical nature."

Of course it is easy enough to imagine circumstances under which reef gold would be subjected to the action of sulphuretted hydrogen or soluble sulphides, seeing how frequently iron pyrites is a constituent of gold reefs, and the above may be a partial explanation of the phenomenon. Skey himself, however, points out in another place that sulphide of gold is decomposed by mercury, so that if all his observations are correct, a film of sulphide of gold should be no obstacle to amalgamation.

Another possible explanation is that the gold has been subjected to severe pressure, and is therefore in an "unannealed" state (see page 24). Prof. Eggleston has pointed out that unannealed gold is only amalgamable with great difficulty (see page 84); of course heating to redness would anneal the gold, and thus account for its amalgamability after calcination.

I myself venture to think that rustiness may very

often, if not in all cases, be due to the existence of gold in an allotropic state, in which it is not affected by mercury; heating to redness would change the gold from this allotropic to its normal condition, and thus render it amalgamable. I shall enter further into this question in Chapter II., page 47.

Whatever then may be the true causes of the phenomena, it must be remembered that gold occurs in reefs in three distinct forms (exclusive of its rare occurrence in chemical combination in the minerals of Class *D*):

I. Ordinary amalgamable gold, usually spoken of as free-milling gold.

II. Gold in some form of intimate physical admixture, or more or less complete chemical combination with sulphides, arsenides, or other compounds of various metals.

III. Rusty gold.

Minerals Mistaken for Gold.—Of the list of minerals given above as liable to occur in gold reefs, a few may at times be mistaken for gold by a careless observer. Thus quartz, and more often mica, stained a bright yellow by oxide of iron, has been mistaken for gold; the extreme hardness of the former, and the highly perfect cleavage of the latter under the knife are quite sufficient to distinguish them from the soft and sectile, but not fissile metal.

Iron and copper pyrites have both frequently been mistaken for gold when tarnished, as they often are. Here again the knife offers a ready means of discrimination, the former mineral being very hard and the latter being scratched with the formation of a gray powder. It is also worth noting that if tarnished pyrites be slowly rotated in the hand, its colour will change with the

varying angles at which light is transmitted through the film of the tarnish, whereas the colour of gold remains unaffected.

Bismuthite, when wet, much resembles gold; in the prospecting pan it appears to have a bright golden yellow colour, and its great specific weight causes it to settle in the place where gold would naturally be looked for. When dry, however, its colour changes at once to a pale greenish yellow. Such mistakes as the above are rarely committed by experienced men, even the most superficial glance serving to detect with certainty the presence of gold. In all cases of doubt the knife should, however, be resorted to, the sectility and malleability of gold serving to distinguish it practically from all other minerals likely to be mistaken for it. Even when it is present in the minutest specks, chemical means need never be employed if the particles are visible under a lens.

To determine whether sulphides or tellurides are auriferous it is absolutely necessary to assay them. The appearance of the mineral affords no guide at all as to whether it contains gold or not. It is usually supposed that fine and close-grained iron and arsenical pyrites are richer in gold than are large and well-marked crystals, and this rule often holds good, although it is by no means universal. When an ore is rich in tellurides, gold may mostly be expected, especially if it comes from an auriferous district, but, as before said, an assay is the only safe or reliable test.

CHAPTER II

PHYSICAL AND CHEMICAL PROPERTIES OF GOLD

Physical Properties

GOLD is certainly dimorphous and may possibly be polymorphous. There are at any rate two well-marked allotropic forms, namely (α) ordinary yellow gold capable of forming crystals, and (β) brown or black, apparently amorphous, pulverulent gold. Of the properties of the latter very little is known, and in the present state of our knowledge we are forced to assume that such of its physical and chemical properties as are not yet known to differ from those of ordinary gold are probably identical with them. It is however highly probable that the falseness of this assumption may be demonstrated before long. I shall here follow the ordinary custom of restricting the use of the word gold without any qualification to the "α" variety, and shall speak of the "β" variety as amorphous gold.

Specific Gravity.—The specific gravity of gold has been variously determined as between 19·30 and 19·34; after melting it is said to be 19·258, which is increased by hammering to 19·367.

Crystallisation.—It crystallises in the cubic system, the

predominating forms being octahedra, dodecahedra, and cubes, or combinations of these forms. Native alloys of gold with silver have also been found crystallised in the same forms as the above.

Malleability.—Of all metals it is the most malleable and the most ductile; it can be beaten into sheets thinner than 0·000004 of an inch, and has been drawn into wire so fine that 600 feet of it only weigh one grain.

If a piece of gold be hammered or rolled out, it at first extends freely and readily; it speedily, however, becomes hard and brittle, and if the operation be continued, can easily be broken into pieces. If, however, at this point the mechanical action be interrupted, and the metal heated to a dull red heat, it at once recovers its original malleability, and can then be further worked until it again becomes brittle, when it can once more be restored to the malleable condition by reheating, and so on, until the limits of extensibility are reached. This operation of restoring malleability by means of heat is known as "annealing." In the unannealed state the free mobility, relative to each other, of the molecules of gold, which constitutes the condition of malleability, is interfered with by these molecules being so driven together as to obstruct each other, each molecule being thus interposed in the path of oscillation of its neighbour. When heated, expansion takes place, the molecules move a little apart and rearrange themselves, and each molecule is then again free to move without being impeded by the adjoining ones, the state of malleability being thus restored.

Tenacity.—The tenacity of gold is by no means great; it is said that a wire 0·08 inch in diameter will support a weight of 150 lbs. Later experiments have determined

the tenacity of gold at seven tons per square inch, with an elongation of 30·8 per cent. on a three-inch test piece. It is classed among the soft metals, being intermediate between silver and tin in this respect; on Mohs's scale its hardness is 2·5 to 3. Its elasticity is very low.

Colour.—It is the only yellow metal known, has a brilliant metallic lustre, and is susceptible of a high polish. In thin sheets it appears of a greenish colour by transmitted light, but is only transparent when less than $\frac{}{0}$ line in thickness. The green light that is transmitted through ordinary gold leaf is turned into a ruby red colour when the thin film of gold is heated to 316° C.; burnishing the leaf will however restore the original green colour. Some observers explain the variations in colour as being optical phenomena exhibited only by gold because no other metal is sufficiently malleable to be obtained in sheets of the required thinness; others consider them an intrinsic property of gold, which would thus be markedly dichroic. It emits a greenish light at the moment of solidification from the molten state.

Fusibility.—It is easily fused in any ordinary fire or before the blowpipe, its melting-point being about 1045° C. (1915° F.); various observers give it as ranging from 1037° to 1381° C. It is thus somewhat less fusible than copper, but far more so than cast iron. It expands markedly when melting, and contracts on solidifying. It is volatile to some extent at very high temperatures, such as those produced by the electric arc or the oxy-hydrogen blowpipe; even a good mouth blowpipe will volatilise appreciable traces of gold, although an intense white heat is necessary to effect this in perceptible quantity.

Conductivity.—It is a good conductor of heat, ranking in this respect next to silver; its conductivity is about

0·532 according to some observers and 0·981 according to others, that of silver being 1. It is also a good conductor of electricity, having an electrical resistance of 1·369 when annealed, and 1·393 in the hard state, annealed silver being taken as 1; its electrical resistance has been determined as 0·8102 microhms per cubic inch for annealed, and 0·8247 microhms per cubic inch for hard drawn gold.

It expands 0·00146 of its volume on being heated from 0° to 100° C.; its specific heat, referred to water as unity, has been variously determined by different experimenters as between 0·0298 and 0·032.

Absorptivity.—Finely divided gold (such for instance as cornets obtained in bullion assays) is capable of absorbing and condensing in its pores small quantities of gases in the following proportions:—

Hydrogen . . .	0·48 of its volume.
Carbonic oxide .	0·29 ,, ,,
Carbonic acid .	0·16 ,, ,,
Air	0·19 to 0·24 (rich in nitrogen).

Chemical Properties

The atomic weight of gold has been variously determined as between 196·33 and 196·85; it is usually taken as 197, which is sufficiently accurate for ordinary purposes; its symbol is Au. It is a triad, and forms two series of compounds, aurous and auric, the oxide types of which are respectively Au_2O and Au_2O_3. It also appears to form an intermediate series of aurylic compounds of the Au_2O_2 type, and compounds corresponding to other stages of basicity are also said to exist, but are by no means well

defined. Gold has but few affinities towards other elements, its most stable compounds being with the haloids; it is affected by very few of the ordinary reagents, and is not attacked by any of the ordinary acids. Its usual solvent is aqua regia (one part of strong nitric and four parts of strong hydrochloric acids). It is, however, important to note that gold, although insoluble in pure nitric acid, is somewhat soluble in it when it contains nitrous acid.

It is also attacked by hydrochloric acid in admixture with any oxidising agent capable of generating chlorine therefrom. It is attacked by selenic acid with the formation of selenious acid. It is not oxidisable in the air at any temperature with or without the presence of moisture. It is attacked by fused nitre, but not by fused potassic chlorate; fused alkalies attack it in the presence of air, an aurate being formed in both of the last-named reactions. It is also attacked by fused alkaline sulphides, a sulphaurate (a fairly stable salt) being formed. The same effect is said to be slowly produced by solutions of ammonic and of alkaline sulphides. Finely divided gold is also soluble in solutions of the alkaline hyposulphites, with or without the addition of cuprous hyposulphite (Russell process). Chlorine attacks it readily in the cold; gold leaf dissolves freely in chlorine water. It also dissolves in bromine. Iodine in the cold has no action on it, but attacks it slowly when exposed to sunlight, or when heated to 50° C. An aqueous solution of hydriodic acid has no effect on it, but it is attacked by an ethereal solution, also by a mixture of hydriodic and sulphuric acids heated to 300° C. Ferric chloride and ferric bromide have no effect on it. Finely divided gold is said to be soluble in ferric sulphate in the

absence of ferrous salts, but this statement requires confirmation. As the result of a series of experiments on this subject, I find that pure sheet gold is quite insoluble in solutions of pure crystallised ferric chloride in distilled water of strengths varying from 10 per cent. to saturation, after a month's exposure to their action. Gold will, however, dissolve in solutions of ferric chloride containing free acid and exposed to the air, or in the presence of an oxidising agent.

Potassic cyanide solution attacks it slowly, more rapidly if finely divided; it seems that this action only occurs in the presence of air, the reaction being probably as follows:—

$$Au_2 + 4KCy + O + H_2O = 2KAuCy_2 + 2KHO.$$

I have found that pure sheet gold will dissolve in solutions of commercial potassic cyanide of strengths varying between 10 and 50 per cent. of the salt, at the rate of from 3 to 5 per cent. in a month. Pieces of pure gold were also treated with pure potassic cyanide dissolved in boiled distilled water, in sealed tubes during one month. In this time, one piece in a solution containing 50 per cent. of the salt lost 0·2 per cent., whilst a similar piece in a 12 per cent. solution lost 4·7 per cent. No gas was evolved during the action. The solubility of gold in potassic cyanide solution was discovered by Bagration in 1843,[1] and he remarks that corrosion is especially active at the surface of the liquid. Elsner seems to have first enunciated the above equation in 1846.[2] The whole subject has received careful investigation more recently from R. C. Maclaurin;[3] he

[1] *Journ. f. Prakt. Chem.* vol. xxxi. [2] *Ibid.* vol. xxxvii.
[3] *Journ. Chem. Soc.* ccclxvi.

comes to the conclusion that cyanates play no part in the reaction, which depends on the direct absorption of oxygen by the solution; he also finds that dilute solutions are more active than concentrated ones, and assigns as the reason the fact, which he has proved, that oxygen is more soluble in the former than in the latter. There is accordingly a certain definite degree of dilution which exerts the maximum solvent effect. Maclaurin confirms the correctness of the reaction given by Elsner. Gold will also dissolve readily in a mixture of solutions of potassic cyanide and ferricyanide, potassic ferrocyanide being formed. The following is probably the reaction :—

$$Au_2 + 2K_3FeCy_6 + 4KCy = 2K_4FeCy_6 + 2KAuCy_2.$$

Gold also dissolves in a mixture of solutions of potassic cyanide and sulphocyanate, and of potassic cyanide and bromocyanogen (Sulman and Teed's reaction); according to the discoverers, the action of this reagent is as follows :—

$$Au_2 + 3KCy + CyBr = 2KAuCy_2 + KBr.$$

Sulphur and sulphuretted hydrogen are usually said to have no effect on gold. Skey[1] says, however, that gold exposed to moist sulphuretted hydrogen, or immersed in a solution of ammonic sulphide, is superficially converted into sulphide without change of colour, and is in this state not susceptible of amalgamation, and this observation has been independently corroborated by Eggleston.[2]

Phosphorus, arsenic, and antimony combine directly

[1] *Chemical News*, vol. xxii. p. 282.

[2] *The Metallurgy of Silver, Gold, and Mercury in the United States*, vol. ii. p. 590. 1890.

with gold under the influence of heat. Mercury combines with it readily at all temperatures (see page 84).

Chlorides.—There exist two well-defined chlorides, aurous chloride, AuCl, and auric chloride, $AuCl_3$. There seems also to be good evidence of the existence of a third intermediate one, aurylic chloride, Au_2Cl_4; higher chlorides are also said to exist, but have scarcely been isolated as yet. These latter compounds have been made the subject of careful study by J. Thomsen.[1]

Aurous Chloride, AuCl.—This is a pale yellow unstable substance, insoluble in water, formed by carefully heating auric chloride to a temperature of 200° C.; heated more strongly, it is resolved into gold and chlorine. It is also produced by the action of water on aurylic chloride, Au_2Cl_4. It is slowly acted on by water in the cold, the action being accelerated by sunlight and becoming rapid at the boiling-point of water, gold and auric chloride being produced.

Caustic potash converts it into aurous hydrate. It combines with alkaline chlorides, forming double salts, the best known of which is potassic aurochloride, $KAuCl_2$; this substance, of a brownish black colour, is produced by gently heating potassic aurichloride, $KAuCl_4$. It is acted on by water, gold being precipitated and potassic aurichloride and potassic chloride being formed.

Auric Chloride, $AuCl_3$, is a deep reddish brown, crystalline substance. It is produced by the direct union of its elements, as by introducing gold leaf into chlorine gas. If a current of chlorine gas be passed over a thin strip of gold heated in a glass tube to 300° C., long, deep red needles of auric chloride sublime and condense on the colder parts of the apparatus, the decomposition of the

[1] *Journal für Praktische Chemie* (2), vol. xiii. p. 337.

auric chloride at this temperature being prevented by the tension of the atmosphere of chlorine. It is usually formed by dissolving gold in aqua regia and evaporating on a water-bath; this leaves a brownish red, deliquescent mass, which is however never pure auric chloride, but always retains some hydrochloric acid, forming hydric aurichloride, $HAuCl_4$. A solution of it, quite free from acid, can only be obtained by decomposing aurous chloride or aurylic chloride with water. It is decomposed slowly by the action of sunlight, and rapidly on being heated. It is soluble in water, alcohol, and ether, the latter removing it entirely from its aqueous solutions.

Its solution is easily decomposed. A stream of hydrogen passed through it throws down metallic gold, and most metals have the same effect. Platinum, however, will not precipitate gold from the solution of its chloride, although it will from a neutral solution of aurichlorides. Tin first throws down metallic gold, which is afterwards converted into purple of Cassius. Phosphorus, arsenic, antimony, and bismuth immersed in a solution of auric chloride become coated with a covering of metallic gold in the cold, and sulphur and selenium on boiling. Carbon precipitates gold in the cold when exposed to light, and so do many native sulphides, such as galena, pyrites, zinc blende, stibnite, cinnabar, &c.; the reaction in the case of galena is:—

$$2AuCl_3 + 3PbS = Au_2 + 3S + 3PbCl_2.$$

Artificial sulphide of copper has a similar action Many organic substances decompose it, especially on heating, and in the presence of caustic potash. Oxalic and formic acids precipitate it; the action takes place in the cold, but it is better to heat the solution to boiling

in order that the precipitation may be quite complete. In the case of oxalic acid the reaction is :—

$$2AuCl_3 + 3H_2C_2O_4 = Au_2 + 6HCl + 6CO_2.$$

Most reducing agents precipitate gold from solutions of its chloride. The most usual precipitant is ferrous sulphate; but any protosalt of iron will have the same effect :—

$$2AuCl_3 + 6FeSO_4 = Au_2 + 2Fe_2(SO_4)_3 + Fe_2Cl_6.$$

Mercurous nitrate, antimonious chloride, and cuprous chloride dissolved in hydrochloric acid precipitate metallic gold.

In a cold solution the finely divided gold remains suspended for a very long time, forming a solution which is brown by reflected and purplish blue by transmitted light; this reaction is perceptible in solutions containing only $\frac{1}{640000}$ of gold, and in the case of such dilute solutions the precipitate only settles with extreme slowness.

Sulphurous, phosphorous and hypophosphorous acids have the same reducing action, as also has arsenious acid, the latter only slowly in the cold, more rapidly on heating.

Stannous chloride containing some stannic chloride produces a purple precipitate known as purple of Cassius; it is a very sensitive reaction, a solution containing only $\frac{1}{4000000}$ of gold becoming purple on the addition of the stannous chloride. This is often used as a test reaction for gold, but it must not be forgotten that silver salts also yield a somewhat similar brownish precipitate, which might be mistaken for purple of Cassius.

Potassic iodide produces in a solution of auric chloride a precipitate of yellow aurous iodide and free iodine.

Argentic nitrate gives a mixed precipitate of argentic chloride and gold oxide, hydrochloric and nitric acids remaining in solution.

Strong sulphuric acid precipitates auric chloride from its concentrated solution, and, on heating, the precipitate decomposes into aurous chloride and chlorine.

Ammonia and ammonic carbonate precipitate the highly explosive ammoniacal oxide of gold known as fulminating gold; the precipitation is not, however, complete.

Caustic potash forms, according to some authorities, a yellowish red precipitate of hydrate, soluble in excess of the precipitant; according to others no precipitate is formed unless there is some organic matter present.

Sulphuretted hydrogen throws down a brownish black precipitate of sulphide, soluble in alkalies and in ammonic sulphide, this being the group-reaction of auric salts.

Sodic phosphate gives no precipitate.

Potassic cyanide forms a yellow precipitate soluble in excess of the precipitant.

Potassic ferrocyanide colours the solution bright emerald green.

Auric chloride is further characterised in the dry state by imparting a bright green colour to a flame, forming a very distinctive spectrum.

Auric chloride forms with most soluble chlorides a very characteristic and interesting series of salts known as aurichlorides.

Hydric Aurichloride, $HAuCl_4$, 3 Aq, crystallises in long square prisms, or in truncated square pyramids; it is a

dark red, deliquescent substance, melts on being heated, and then decomposes, giving off chlorine and hydrochloric acid.

It is produced by dissolving gold in aqua regia containing but little nitric acid and heating till this is all completely decomposed, when the double salt crystallises out on cooling.

Auric chloride as ordinarily prepared always contains more or less of this salt.

Ammonic Aurichloride, NH_4AuCl_4, 3 Aq, forms yellow transparent needles. Aurichlorides of potassium (with varying amounts of Aq), sodium (with 2 Aq), lithium, magnesium (with 7 Aq), barium, calcium (with 7 Aq), cobalt, nickel, manganese, zinc, cadmium, and silver are known; they all form reddish to greenish yellow salts, crystallising very readily with various amounts of water of crystallisation.

Aurylic Chloride, Au_2Cl_4, has been produced by treating amorphous gold with chlorine. It is a hard, dark red, friable substance, hygroscopic, decomposed by water into auric and aurous chlorides.

Aurous Bromide, AuBr, is a gray, unctuous substance, insoluble in water, produced by cautiously heating auric bromide; water decomposes it into gold and auric bromide.

Auric Bromide, $AuBr_3$.—Gold dissolves slowly in bromine water; on evaporating, a blackish gray residue of auric bromide is left, capable of crystallising in scarlet crystals. The solution is bright red, and possesses considerable colouring power. It is also produced by the addition of a solution of hydrobromic acid to a solution of auric chloride, the solution becoming dark red, and hydrochloric acid being produced; the auric bromide

can be separated by the addition of ether, in which it is highly soluble and which removes it completely from the aqueous solution.

The affinity of auric oxide for hydrobromic acid is markedly greater than for hydrochloric acid.

The reactions of auric bromide are like those of auric chloride. Auric bromide forms a series of double salts with soluble bromides, many of which have been isolated and studied.

Hydric Auribromide, $HAuBr_4$, is a dark red crystalline body.

Auribromides of potassium, sodium, magnesium, barium, zinc, &c., have been produced; they are all red or reddish brown and highly crystallisable.

Aurylic Bromide, Au_2Br_4, has been produced by the action of bromine on finely divided amorphous gold.

Aurous Iodide, AuI, is the most stable of the iodides of gold, and is produced in all reactions of iodine with gold.

It is formed by adding potassic iodide to auric chloride:—

$$AuCl_3 + 3KI = AuI + 3KCl + I_2.$$

Excess of potassic iodide must be avoided, as this decomposes aurous iodide, and the iodine produced must be volatilised at the lowest possible temperature. The salt forms a greenish yellow, crystalline powder, insoluble in water; it splits up into gold and iodine when heated above 60° C. Most reagents decompose it.

Auric Iodide, AuI_3, is a very unstable body, that splits up into aurous iodide and iodine almost as soon as formed. When a solution of auric chloride is added to potassic iodide, a dark green precipitate is produced, soluble in potassic iodide, with the formation of potassic auri-iodide:—

$$4KI + AuCl_3 = KAuI_4 + 3KCl.$$

When not dissolved in excess, the precipitate cannot be dried without decomposition.

Several auri-iodides of the general formula $MAuI_4$ are known; they form blackish, generally prismatic crystals, and are very unstable.

Fluoride of Gold is said to exist, but has never been isolated; nothing is known of its properties.

Cyanides of Gold.—Two are known, aurous and auric, the latter in combination only; they are of considerable practical importance, being largely used in electrogilding, and in the extraction of gold on a large scale.

Aurous Cyanide, AuCy, is a yellow, granular, crystalline substance produced by heating potassic aurocyanide with hydrochloric or nitric acid. It is insoluble in water and acids, being attacked by aqua regia only. On heating, it is split up into gold and cyanogen. It is soluble in ammonia and in sodic hyposulphite, also in potassic cyanide. A cold solution of caustic potash has no action on it, but on boiling, potassic auricyanide and metallic gold are produced.

It forms double salts with soluble cyanides, known as aurocyanides, the most important being the potassium compound, $KAuCy_2$. This appears to be produced when gold leaf is treated with a solution of potassic cyanide in the presence of air. (See page 28.)

It is usually produced by dissolving fulminating gold in potassic cyanide. Ammonia is added to a solution of auric chloride and the resulting precipitate washed and treated with potassic cyanide.

A solution of potassic aurocyanide dissolves iodine, producing a dark brownish violet, readily crystallisable substance, potassic auri-iodo-cyanide, $KAuCy_2I_2$.

Analogous compounds with chlorine and bromine are also known.

Potassic Aurocyanide is decomposed by most mineral acids, these precipitating aurous cyanide and liberating hydrocyanic acid; it is very extensively used in electro-gilding, &c.

Auric Cyanide, $AuCy_3$, exists only in double salts known as auricyanides.

Potassic Auricyanide, $2KAuCy_4$, 3Aq, is prepared by mixing hot concentrated neutral solutions of auric chloride and potassic cyanide. It forms colourless tabular crystals, soluble in warm water and in alcohol; these give up two molecules of water of crystallisation in dry air, but the third is only expelled at a temperature of 200° C. At a low red heat the salt is fusible; slight decomposition sets in about 300° C., but this is not complete even at a high temperature.

Chlorine and bromine have no effect upon it. Iodine dissolves in a hot solution of the salt, displaces two molecules of cyanogen, and produces potassic auri-iodo-cyanide, $KAuCy_2I_2$, Aq, which crystallises in thin, brown, acicular crystals; corresponding chloro- and bromo-salts are known to exist, as also are auricyanides of hydrogen (with 6 Aq), ammonium, sodium, barium, calcium, strontium, and zinc.

Oxides of Gold.—Two well-defined oxides are known to exist, aurous and auric oxide, Au_2O and Au_2O_3; there seems also to be an aurylic oxide, Au_2O_2, and an even higher stage of oxidation, Au_2O_5, is recognised by some authorities; its existence is, however, doubtful.

Aurous Oxide, Au_2O.—This is best prepared by adding a solution of mercurous nitrate to a neutral solution of auric chloride, avoiding excess of the precipitant :—

$$2AuCl_3 + 2Hg_2(NO_3)_2 + H_2O = Au_2O + 3HgCl_2 + Hg(NO_3)_2 + 2HNO_3.$$

Thus prepared it forms a dark purplish precipitate. It is also produced by the action of organic matter on potassic rate, and also by the prolonged boiling of auric hydrate with caustic potash, but these methods do not produce it in a state of purity.

When dried it forms a purplish blue powder, which decomposes at 250° C. into gold and oxygen.

Hydrochloric acid acts upon it, producing gold and auric chloride :—

$$3Au_2O + 6HCl = 2AuCl_3 + 2Au_2 + 3H_2O.$$

When freshly precipitated it is soluble in alkalies, forming aurites of the general formula MAuO ; potassic aurite, KAuO, decomposes on standing, gold being precipitated.

Auric Oxide, Au_2O_3.—This is best known in the hydrated state, its existence in the anhydrous condition having long been considered doubtful. It has, however, been prepared as a blackish brown powder by heating the hydrate cautiously to 100° C. ; above this temperature it is decomposed into gold and oxygen ; it is an extremely unstable compound.

It is prepared by adding auric chloride to a solution of caustic potash, and decomposing the solution thus formed with sulphuric or nitric acid. The resulting precipitate may be purified by dissolving in concentrated nitric acid ; on diluting this solution with water, auric hydrate is reprecipitated. Its formula is given variously as $Au_2H_6O_6$ and $Au_2H_2O_4$. It forms a brownish black powder, and is a very unstable body, readily reduced to the metallic

state by most reducing agents. It is soluble in concentrated sulphuric and nitric acids; hydrochloric and hydrobromic acids dissolve it, forming the corresponding haloid salts; hydriodic acid decomposes it, producing metallic gold; hydrofluoric acid is said by some authorities not to affect it; according to others it dissolves it.

Ammonia forms with it the highly explosive compound known as fulminating gold.

Alkalies dissolve it, forming salts known as aurates, having the general formula $MAuO_2$.

The potassium salt, $KAuO_2$, 3 Aq, forms pale yellow crystals, very soluble in water; acids decompose its solution, separating auric hydrate. This salt is formed when nitre is fused with finely divided gold. It is also produced when auric hydrate is boiled with potassic chloride:—

$$Au_2H_6O_6 + 2KCl = 2KAuO_2 + 2H_2O + 2HCl.$$

With most metallic salts potassic aurate gives a precipitate of an aurate soluble in excess.

A solution of potassic aurate is sometimes used in electrogilding.

Aurylic Oxide, Au_2O_2, has been described by Berzelius and Prat; its definite existence is still, however, somewhat doubtful.

Oxysalts of Gold

Several of these have been described, but owing to the low chemical affinities of gold they are all very unstable, and their preparation and investigation are attended with great difficulties.

Auric Hydric Nitrate, $AuH(NO_3)_4$, 3 Aq, is produced when auric hydrate is dissolved in strong nitric acid and

the solution evaporated *in vacuo* at 60° C. Crystals are thus produced melting at 72° C., forming a basic nitrate at 100° C., and completely decomposed at 180° C.

Water decomposes its solution, forming auric hydrate.

Sulphates of Gold.—Both aurous and auric sulphates seem to exist; the former is said to have been obtained in crystals forming small red prisms. The latter is known only in solution in concentrated sulphuric acid; the solution is decomposed on dilution, auric hydrate being precipitated.

Auric Sulphite has not been isolated, but is known as a double salt combined with sulphites of the alkalies.

Potassic Auric Sulphite, $K_{10}Au_2(SO_3)_8$, 5 Aq, is produced by adding potassic sulphite to a solution of potassic aurate with a slight excess of caustic potash, the solution depositing fine acicular crystals of a yellow colour. It is insoluble in alkaline solutions and very soluble in water, whence it may be precipitated by the addition of alcohol in very fine white needles, which take a yellow tinge on desiccation. It is decomposed by heat or by acids.

Auric Hyposulphite is also only known as a double salt with alkaline hyposulphites.

Sodic Auric Hyposulphite, $Na_3Au(S_2O_3)_2$, 2 Aq, is prepared by precipitating a mixture of auric chloride and sodic hyposulphite solutions by means of alcohol. It is a comparatively stable salt, soluble in water; its solution gives reactions neither of gold nor of hyposulphurous acid. It is sometimes used in photography.

Carbonate of Gold is not known to exist.

Silicate of Gold.—When finely divided gold or a salt of gold such as the chloride is mixed with silica and alkalies or alkaline earths and fused, as in glass-making, a yellow silicate of gold or an aurosilicate of the bases present

is produced; when this glass is heated to redness in the air, it assumes a beautiful purple shade, the exact cause of which has not yet been determined. Alkaline aurosilicates seem also to be produced by mixing solutions of alkaline aurates and alkaline silicates.

Purple of Cassius.—This substance was first discovered by Cassius of Leyden in 1683. Its constitution has been variously explained by different chemists, the two most important and most plausible views being that it is either an aurous stannate, or else a "lake" of stannic hydrate coloured by finely divided gold. Figuier, following the former hypothesis, gives its formula as $Au_2O(SnO_2)_3 + 4H_2O$. It is produced by adding a mixture of stannous and stannic chlorides to auric chloride solution, when a fine purple precipitate is produced; stannous chloride alone gives a brown precipitate. It is also formed by the action of metallic tin on a solution of auric chloride, and in various similar methods, the object of all of which is to obtain the precipitate in a fine state of division.

Heated to 100° C. it retains water; at a higher temperature it gives off water, but not oxygen, and becomes brick red. It yields no gold up to mercury when triturated with it. It is used in the arts for colouring glass, melted with which it produces different shades of red varying from rose pink to a deep ruby red.

Sulphides of Gold.—Gold has but little affinity for sulphur, and it is as yet by no means certain what compounds of these elements really exist; aurous and auric sulphides are usually described as having been isolated; both are blackish brown in colour.

Aurous Sulphide, Au_2S, is said to be produced by passing sulphuretted hydrogen through a boiling strong solution of auric chloride, and *auric sulphide*, Au_2S_3, by

passing the gas through a cold dilute solution. It appears to be impossible to produce definite compounds of gold and sulphur by passing sulphuretted hydrogen into a solution of auric chloride under any circumstances; the precipitate seems always to contain free sulphur in addition to the sulphides, and perhaps, under some conditions, free gold also. The precipitation is, however, quite complete, and this is the group-reaction used to separate gold in the course of wet analysis. Auric sulphide is attacked by potassic cyanide. It is also decomposed by metallic mercury forming mercuric sulphide and liberating gold. It is insoluble in hydrochloric or in nitric acid, but is dissolved by aqua regia. It is also soluble in ammonic sulphide, in caustic alkalies, and in alkaline sulphides.

Auric sulphide is readily decomposed at low temperatures, but forms with alkaline sulphides a series of sulphaurates, which are more stable.

Sodic Sulphaurate, $NaAuS_2$, 8Aq, is produced by dissolving precipitated auric sulphide in sodic sulphide; it crystallises out in colourless hexagonal prisms. It is also produced by heating to redness a mixture of finely divided metallic gold, sodic sulphide, and sulphur.

A double sulphide of gold and silver, $2Au_2S_3$, $5Ag_2S$, has been produced by passing sulphur vapour through a molten alloy of gold and silver under a layer of borax. It is crystalline and brittle, has a sp. gr. of 8·159, is unchanged when heated in the air, but is decomposed by nitric acid.

Sulpharsenates of Gold.—A pyrosulpharsenite, $Au_4As_6S_{15} = (Au_2S_3)_2(As_2S_3)_3$, and a sulpharsenate, $AuAsS_4$, have been isolated.

Auric Selenide, Au_2Se_3, and *Telluride*, Au_2Te_3, are formed by precipitating a solution of auric chloride with

seleniuretted and telluretted hydrogen respectively. A sulphotelluride, Au_2TeS_5, has been produced by precipitating a solution of auric chloride by potassic sulphotelluride.

Gold and Nitrogen.—No direct compound of these elements has been isolated, but it enters into the composition of so-called fulminating gold. This is produced when ammonia is added to auric chloride or hydrate; its composition is said to be $2(NH_3AuN), 3H_2O$. It can be handled when moist, but when dry the slightest physical cause is capable of determining its explosion with great violence. When it is formed by the addition of ammonia or ammonic carbonate to solution of the chloride, the resulting precipitate appears to contain chlorine:—

$$4(NH_3AuN) + 4NH_3AuCl,9H_2O.$$

This compound is also fulminating. When auric hydrate is digested in ammonic sulphate, sulphuric acid is liberated and fulminating gold produced. It is an olive-green compound, which with great care can be dried at 100° C. When dry it explodes with great violence by the gentlest friction, by contact with any hard body, by the action of electricity, or by heating to 140° C. Its preparation is dangerous in the extreme, and in fact neither ammonia nor any salt of ammonia can be added to a gold solution without the risk of accident, unless great care be taken that the resulting compound shall never be allowed to become dry.

Gold and Phosphorus.—When finely divided gold is heated with phosphorus out of contact with air, combination ensues, phosphides of gold being formed. The formulas AuP and Au_2P_3 have been assigned to this

body, and it is quite possible that there may be several phosphides. The phosphide is described as a gray metallic body of sp. gr. 6·6, losing phosphorus when strongly heated. It is not attacked by hydrochloric acid, but is decomposed by nitric acid, metallic gold being liberated.

Gold and Arsenic.—A coarse-grained, easily fusible, gray arsenide of gold is said to be produced when finely divided gold is heated in arsenic vapour (see page 76).

Gold and Silicon.—Gold melted with silicon will combine with it; with 3 per cent. of silicon the compound is of a dirty yellow colour and rather brittle; with 10 per cent. yellowish gray and brittle, and with 20 per cent. gray and very brittle. These are possibly definite silicides of gold diffused through an excess of metallic gold.

Gold and Carbon are not known to combine.

Allotropic Gold (β Gold)

An allotropic modification of gold appears to be produced when a very dilute solution of gold chloride is precipitated by means of a reducing agent, such as ferrous sulphate, oxalic acid, &c. The same substance is perhaps also produced when an alloy of gold, with silver or copper for instance, is acted on by nitric acid, the other metal being dissolved thereby and this form of gold remaining behind. When the alloys of gold with sodium or potassium are thrown into water they are decomposed, the gold being left behind as a fine blackish brown powder, which appears to be an extreme form of this allotropic variety. It forms an amorphous, incoherent, brown powder, having no metallic lustre. Rose has found that the specific gravity of amorphous gold when precipitated from a solution of the chloride by ferrous sulphate varied from

19·55 to 20·72, and when precipitated by oxalic acid was 19·49. I have found the specific gravity of the amorphous gold left behind when the silver is dissolved out from a silver-gold alloy to be 19·51,[1] whilst gold similarly left on dissolving out the base metal from alloys with lead and copper has a specific gravity that may rise to 19·78, or in all cases higher than that of ordinary melted gold. It has been said by some observers that gold when precipitated from its solutions is crystalline, but my own observations are opposed to this as a universal statement. Gold when thrown down from moderately strong solutions has an appearance suggesting crystalline structure, but when deposited from dilute solutions it seems under the microscope to form amorphous irregular granules with no indication of crystalline arrangement. When finely divided gold, as precipitated on a photograph in the process of "toning," is examined under the microscope no crystalline structure can be distinguished in the particles of gold. At my request these observations have been repeated by my friend Mr. F. E. Lott, A.R.S.M., &c., who finds that the employment of gold solutions ranging in strength from 10 per cent. to 0·0001 per cent. makes no difference in the form of the precipitate, which under 500 diameters and even under 800 (immersion) has no trace of crystalline appearance, nor do the particles show any tendency towards regular arrangement. Beyond this nothing is known of its physical properties. It is immediately converted into α gold by heating to a temperature which I have determined to lie between 400° and 500° C., when the loose pulverulent mass shrinks, becomes

[1] The specific gravity of this gold after "annealing" was 18·728, and after melting 19·186. See *Trans. Am. Inst. Min. Eng.*, 1894, xxiv. p. 705.

coherent, and assumes the characteristic yellow colour and metallic lustre of α gold. The same effect is wholly or partially produced by violent percussion or friction.

Its chemical properties are usually supposed to be identical with those of α gold. Thomsen has, however, found that when β gold is acted on by chlorine or bromine, aurylic salts are formed instead of auric; I have also found a very important difference in one respect, which had previously been noted by Knaffl, who states that gold, precipitated by ferrous sulphate, is difficult of amalgamation.[1] Pure gold (see page 84) readily combines with mercury to form amalgams; β gold obtained by precipitation is, however, only with great difficulty attacked by mercury, not at all when moist, and very slightly after drying on an air bath; when the mercury is heated to its boiling-point amalgamation is slightly promoted, but is still far from complete; the addition of a small fragment of sodium to the mercury has no effect unless a few drops of water are also added, when complete amalgamation at once ensues. In this connection I may also refer to the action of mercury upon purple of Cassius (see page 41). This substance has been by many chemists supposed to consist of a "lake" of stannic hydrate, throughout which free metallic gold is suspended, and one of the main arguments against this view is its indifference to the action of mercury; if, however, this gold be supposed to be of the β allotropic variety, as is probable from its colour, its indifference to the action of mercury is quite intelligible.

Thomsen in his thermochemical investigations[2] has pointed out that gold precipitated by sulphurous acid

[1] *Dingl. Poly. Journal*, 168–282.

[2] *Journal f. Prakt. Chem.* xiii. p. 348.

from a solution of its chloride forms a coherent, light-coloured powder, whereas the same reagent precipitates it from its bromide solution as a very fine, dark-coloured powder, which obstinately retains its pulverulent form, and further that these two are allotropic modifications, as shown by the fact that each atom of the latter has a thermic energy of 3210 *cal.* more than the former. It need hardly be said that this forms a strong argument in favour of the allotropism of gold.

There is no reason why β gold should not occur in nature as well as the α variety, and it is quite possible that the above-stated indifference of β gold to mercury may account, if not wholly, at any rate partly, for the phenomenon known as the "rustiness" of gold, a point which has already been referred to and will be again further on.

CHAPTER III

PHYSICAL AND CHEMICAL PROPERTIES OF MERCURY

Use of Mercury.—Every system of gold milling, properly so called, depends upon the employment of mercury for the extraction of gold from the ore. This substance being the solvent mainly employed for the separation of gold, a knowledge of its physical and chemical properties is indispensable in order that the science of gold milling may be thoroughly understood.

Physical Properties of Mercury

Mercury is a metal which differs from all the other common metals in that it is liquid at ordinary temperatures, its melting-point being below the usual temperature of the air.

Specific Gravity.— Its specific gravity when liquid has been variously estimated at between 13·589 and 13·613, one of the latest observers giving it at 13·5953 at 0° C.; the specific gravity of solid mercury is given as between 14·193 and 14·391.

Its vapour density is 6·976.

Thermal Data.—Its solidifying point is usually taken at $-40°$ C.; more accurately it appears to be $-39°·44$, whilst solid mercury commences to melt at $-38°·85$ C.

Its boiling-point is given as between 357°·25 and 360° C.; mercury emits vapour however at 15° C., and perhaps even at lower temperatures; so that, if a little mercury be placed in the bottom of a bottle and a piece of gold suspended in the upper part of the same bottle, the gold will in time be amalgamated and whitened by the mercurial vapours at ordinary atmospheric temperatures.

Its coefficient of expansion is 0·00018 for each degree between 0° and 100° C., increasing slightly with the temperature.

Its specific heat is 0·03247 in the solid state, 0·03312 in the liquid state up to 20° C., and 0·03278 above this temperature. Its latent heat of fusion is 2·84 calories, or about eighty times as much as is required to raise the temperature of solid mercury 1° C. It is the worst conductor of heat of all the metals, its conductivity being only 5·33, that of silver being taken at 100.

Its electrical conductivity is even lower, being only 1·63, that of silver being taken at 100; its electrical resistance per cubic inch is 37·15 microhms.

Optical Properties.—Mercury is pure white in colour, opaque, with a highly brilliant surface; its vapour is transparent and colourless.

Structural Properties.—Solid mercury crystallises in octahedra; it is soft, sectile and malleable, having a very low tenacity. In all these qualities it much resembles lead. Liquid mercury does not adhere to any substance except such metals as it readily amalgamates with under ordinary circumstances; it "wets" such metals, forming a thin adherent film on them, but when placed on other substances to which it does not adhere, its surface tension comes into play, and it forms spherules or takes a convex surface. It is only pure mercury that forms an approximately true

sphere; for when it contains impurities in the form of foreign metals it loses its sphericity, and the globules elongate, forming a so-called "tail." Allowing a minute particle of mercury to run down a gently inclined plane of glass forms therefore a rough test of its purity, as the presence of foreign metals causes the globules to assume a pear-like shape; if it contains $\frac{1}{4000}$ part of lead, it will adhere in the form of a thin film to the glass plate.

Flouring of Mercury.—When mercury is vigorously agitated with some saline solutions such as a solution of calcic chloride, with fatty substances, oils, turpentine, ether, sugar, sulphur, even with water (and even to some extent by itself *in vacuo*), it is split up into a number of minute globules, invisible to the naked eye, forming a soft gray powder. This phenomenon is known as the "flouring" or "sickening" of mercury. When the foreign body which has caused this splitting up is removed, the particles tend partially to reunite so as to reproduce ordinary mercury. The phenomenon of flouring is by no means thoroughly understood; for a long time the gray powder was looked upon as a suboxide of mercury, but it appears to be certain that most of the substance is metallic mercury, although it is very probable that the minute globules are covered with a film of oxide (or in some cases sulphide) of mercury. The substances that act most energetically in producing flouring are vegetable or animal fats, grease, oil, &c. Thus good mercurial ointment, which consists of metallic mercury rubbed up with lard, contains the mercury uniformly distributed through its mass in the form of microscopic globules from $\frac{1}{500}$ to $\frac{1}{1000}$ of a line in diameter. It is these minute globules that cause the grey or bluish colour of the ointment. Oil causes flouring with great ease, but mineral far less so than

vegetable or animal oil. All unctuous substances, even such as clays, also promote this effect, which appears to be a result of the physical presence of the body determining it, and not a chemical action. Reducing agents appear to promote to some extent the coalescence of the particles of floured mercury; this effect is energetically produced by an electric current, when the negative electrode is placed in the floured mercury, and the positive in a little acidulated water which forms a layer above the mass of floured mercury.

The same power is exhibited by a particle of sodium in the presence of water. This effect is in each case attributed to the vigorous reducing action of hydrogen in the nascent state, which is evolved in contact with the mercury.

Chemical Properties of Mercury

General Chemical Properties.—Pure mercury is not affected by exposure to air, either dry or moist, at ordinary temperatures; when impure, its surface becomes covered by a pellicle said to consist of a mixture of metallic mercury and mercuric oxide. When heated in air to 350° C., just below its boiling-point, it is oxidised superficially. Water agitated with mercury takes up a little of it, acquirin a distinct metallic taste; it appears not to be dissolve strictly speaking, but to be taken up mechanically in a state of minute subdivision.

Sulphur combines directly with mercury both in the form of vapour and by rubbing the two substances together. Mercury vapours at the ordinary temperature are absorbed by sulphur.

The haloid elements, chlorine, bromine, and iodine, unite directly with mercury.

Dilute sulphuric acid does not attack mercury, but it is dissolved by hot strong acid with the evolution of sulphurous acid.

Hydrochloric acid has no action on mercury; according to some authorities, however, chloride of mercury is formed in the presence of air. Hydrobromic acid is slowly, and hydriodic acid readily, decomposed by it.

It is oxidised by a solution of potassic permanganate, forming mercurous oxide in the cold, and mercuric oxide when heated.

Its symbol is Hg; atomic weight 200, and vapour density (compared with hydrogen as unity) 100, so that its atom occupies two volumes. It is diatomic, and forms two well-marked series of compounds, mercurous and mercuric, in the former of which it acts as a monad, in the latter as a dyad.

Mercurous chloride, Hg_2Cl_2, and mercuric chloride, $HgCl_2$, may be taken as types of these two classes of compounds.

Amalgams.—Mercury has a strong affinity for most of the metals, and readily forms alloys with them, which are known by the generic term of amalgams. This term, which has become consecrated by use since the time of the early alchemists, is a bad one in so far as it implies that the alloys of mercury with the metals differ intrinsically from the alloys of any other metals. It is merely the, so to speak, accidental fact that the melting-point of mercury is below ordinary temperatures that causes combination with other metals to take place very readily; but the action is just the same as when a piece of platinum, for instance, is immersed in molten lead, when a lead-platinum alloy forms at a temperature far below the melting-point of the platinum.

Amalgams of all metals have certain properties in common. They are all white; when the proportion of mercury is low, they tend to form solid crystalline bodies; as the proportion increases, the amalgams become pasty, and finally liquid. All amalgams appear to be soluble to some extent in pure mercury, and are miscible with mercury in all proportions. When such a mixture is subjected to filtration, more or less solid amalgams, having at times approximately constant compositions, are left behind, and the excess of mercury is separated, being, however, saturated with as much of the amalgam as it is capable of holding in solution.

Amalgams are formed in various ways; very many, such as those of potassium, sodium, gold, silver, copper, zinc, &c., are produced by direct union of the metals at ordinary temperatures, favoured sometimes by gently heating. Some can be formed by electrolysis of a salt of the metal, mercury forming the negative electrode—iron amalgam can be produced in this way—or else by the action of sodium or ammonium amalgam on solutions of the metallic salts. In this way amalgams of platinum, iron, and aluminium, which cannot be formed by direct combination, are produced.

As a rule the formation of amalgams causes an absorption of heat with corresponding lowering of temperature; a few metals, such as potassium, sodium, and cadmium, cause, however, a rise of temperature on amalgamating.

Most of the ordinary metals unite with mercury, but platinum, iron, aluminium, chromium, manganese, nickel, and cobalt will not unite directly; their amalgams can, however, be produced by the two latter methods mentioned above.

Mercury and Antimony.—These metals do not unite in

the cold, but on heating a soft amalgam is produced, from which antimony gradually separates out as a black powder.

Mercury and Arsenic.—Arsenic heated with mercury forms a gray compound of 5 parts of mercury to 1 of arsenic. When sodium amalgam is rubbed up with moistened arsenious acid, arsenic separates as a blackish powder, but does not combine with the mercury. Several double sulphides of mercury and arsenic are known.

Mercury and Tellurium form a tin-white granular amalgam.

Mercury and Bismuth amalgamate directly at ordinary temperatures, forming a white granular crystalline alloy.

Mercury and Zinc form a white crystalline amalgam.

Mercury and Cadmium amalgamate very readily, forming silver-white crystalline alloys; their employment has been proposed in gold amalgamation in the place of sodium amalgam.

Mercury and Tin unite very readily, forming silver-white alloys, which are largely used for "silvering" mirrors.

Mercury and Lead amalgamate very easily, forming a tin-white granular alloy.

Mercury and Copper combine freely. The amalgam is silver-white, crystalline, and when well squeezed contains 4·77 parts of mercury to 1 of copper ($=Cu_2Hg_3$); 1,000 parts of mercury will dissolve 0·04 parts of copper.

Mercury and Silver combine in the cold, but more readily when heated. A beautiful crystalline amalgam is produced when a globule of mercury is left in a solution of argentic nitrate. Many definite solid crystalline amalgams of silver are known, some occurring as natural minerals.

Mercury and Gold amalgamate readily (see page 84).

Mercury and Iron do not combine directly, but iron can be amalgamated by means of sodium amalgam. Amalgams of iron, containing less than 1·5 parts of iron to 100 of mercury are fluid, up to 11 parts of iron pasty, and above that solid and fusible. When iron amalgam is exposed to the air, a coating of grayish black oxide of iron forms on it, but the amalgam is decomposed completely only after standing some considerable time. When it is heated in the air, the iron burns, throwing out bright scintillations.

Mercury and Sodium combine energetically, their combination being attended by the evolution of both heat and light, the maximum effect being produced when the elements are present in such proportions as to form the solid amalgam $Hg_{12}Na_2$. Sodium amalgam is best prepared by heating pure dry mercury gently in a flask or basin, and throwing in small, dry, clean chips of sodium freshly cut from a lump of the latter metal, one piece at a time, until sufficient has been introduced. The combination with each fragment is attended with incandescence. Sodium amalgam is as white as mercury and as brilliant.

When containing 30 parts of mercury to 1 of sodium it is rather hard under the file, and has a crystalline, foliated structure. With less mercury than the above it is highly crystalline and brittle. It forms long prismatic crystals, that can be freed from adhering mercury by pressure. According to Berzelius it crystallises in cubes. With 40 parts of mercury to 1 of sodium it is still solid, but softer. With 60 parts of mercury it forms a mass which is a stiff paste at 21° C., and seems to be composed of confused interlacing crystals. With 80 parts of mercury it forms a thin paste showing numerous granular

crystals. With 100 parts of mercury it forms a thick fluid, which seems to be partly solid and partly fluid. With 130 parts of mercury it is quite fluid.

It is said that when an amalgam containing 3 per cent. of sodium is allowed to stand under water, needles having the formula $Hg_{12}Na_2$ separate out.

When gently heated, solid sodium amalgam melts; it gives off its mercury at a heat below redness. In air it decomposes slowly, more rapidly under water; the solid amalgams high in sodium decompose in water more rapidly than those low in sodium. The ultimate products of the decomposition are mercury and sodic hydrate. Sodium amalgam can only be preserved in air-tight vessels.

Mercury and Potassium.—Potassium amalgam is similar in properties to the last. The evolution of heat when the elements combine, here reaches its maximum, when their proportion is such as to produce an amalgam of the composition $Hg_{24}K_2$.

Haloid Compounds

Bromides

Mercurous Bromide, Hg_2Br_2, is white, insoluble in water, fusible and volatile below redness; it is formed by the action of mercury on mercuric bromide, or by precipitating a soluble mercurous salt by means of a soluble bromide.

Mercuric Bromide, $HgBr_2$, is formed by the action of excess of bromine on metallic mercury. It is soluble in water, alcohol, and ether; it is easily decomposed by nitric and by sulphuric acids. It forms double salts with other

bromides, which are soluble in water. Of these mercuric potassic bromide, $HgBr_2 2KBr$, may be taken as the type.

Iodides

Mercurous Iodide, Hg_2I_2, is produced by grinding up 200 parts of mercury and 127 of iodine with a little alcohol till all the mercury has disappeared, forming a greenish paste; this is thoroughly rubbed up with alcohol, washed with alcohol to remove mercuric iodide, and dried.

It is also formed by adding an alkaline iodide to a solution of a mercurous salt; excess of the iodide must be avoided, because alkaline iodides split up mercurous iodide into mercuric iodide and free mercury.

It is a dark yellowish green powder, apt to decompose slowly when dry into mercury and mercuric iodide; this decomposition is accelerated by the presence of numerous reagents. It is soluble in ammonia.

Mercuric Iodide is formed by rubbing up mercury with iodine in excess, or by precipitating a solution of a mercuric salt with potassic iodide. The precipitate is soluble in excess of either reagent. It is markedly dimorphous, there being a red and a yellow form, the former being the more stable. The red form is produced by precipitation as above, the yellow by fusion or sublimation. It is not very soluble in water, but dissolves in many acids and in solutions of ammoniacal and other neutral salts, such as sodic and potassic chlorides, &c.; it is soluble in alcohol. It is soluble in solutions of many metallic iodides, producing double salts of the general formulas HgI_2MI and $HgI_2 2MI$, the latter being always the more stable form. With dyads, compounds of the type HgI_2DI_2 are produced.

Fluorides of mercury are known to exist.

Chlorides

These are the most important compounds of mercury, and both of them are used in the arts.

Mercurous Chloride, or calomel, Hg_2Cl_2, is a white amorphous body, insoluble in water, volatile, subliming in square prismatic crystals. It can be made by the direct action of chlorine gas on excess of mercury, by the action of ferric chloride on mercury, and most commonly by the distillation of a mixture of salt and mercurous sulphate, when double decomposition ensues. However prepared, it must be carefully washed with water to remove any mercuric chloride that may have been formed, and which is soluble in water. Prolonged boiling with water forms a little mercuric chloride, liberating mercury. The action of hydrochloric acid forms mercuric chloride. Most oxidising agents transform it into mercuric chloride, and most reducing agents into mercury.

Heated with sulphur, mercuric chloride and sulphide are formed :—

$$Hg_2Cl_2 + S = HgS + HgCl_2.$$

With sulphide of antimony, chloride of antimony and mercuric sulphide are produced.

Chlorine and aqua regia convert it into mercuric chloride.

Mercuric Chloride, or corrosive sublimate, $HgCl_2$, is formed by the direct action of chlorine in excess upon mercury, or usually by distilling together mercuric sulphate, salt, and manganic dioxide. It sublimes in white crystalline masses, and is readily soluble in water, alcohol, and ether. Hot hydrochloric acid dissolves it freely, the mass solidifying on cooling. Hot sulphuric acid only attacks it very slowly.

It has an acrid metallic taste, and is a very violent poison.

It is easily reduced by reducing agents to mercurous chloride, or to metallic mercury. Most metals reduce it.

Solutions of caustic alkalies in small quantities give a reddish brown precipitate of oxychloride, in larger quantities an orange yellow precipitate of mercuric oxide. Carbonates give a precipitate, which is at first white but soon changes to the reddish brown oxychloride.

With albumen it forms insoluble compounds.

It is very largely used as an antiseptic.

Oxides

There are only two known: mercurous and mercuric.

Mercurous Oxide, Hg_2O, is a very unstable body; it is produced as a brownish black powder by the action of alkalies on mercurous chloride. The precipitate must be washed and dried at a low temperature, and not exposed to light; even diffused light decomposes it into mercuric oxide and metallic mercury.

Mercuric Oxide, HgO.—This oxide is produced when metallic mercury is maintained at a temperature just below its boiling-point for some considerable time in air or oxygen. A higher temperature decomposes it again into metallic mercury and oxygen. It may also be prepared by adding alkalies to a solution of mercuric chloride, when it is precipitated in the anhydrous state as a reddish yellow powder. It appears to be dimorphous, a red and a yellow variety being known to exist. It is very slightly soluble in water, to the extent of 1 part in 200,000; even this small amount gives a distinct metallic taste. It is capable of forming compounds with the haloid salts of mercury. At ordinary temperatures it is a very stable compound.

Sulphides

Mercurous Sulphide, Hg_2S, is an unstable black substance produced by the action of alkaline sulphides on mercurous salts; it readily splits up into mercury and mercuric sulphides, and may be only an intimate mixture of these two substances.

Mercuric Sulphide, HgS, is markedly dimorphous, there being both a black and a red variety; the latter is known as cinnabar or vermilion, the former name being applied to the native mineral, which forms the only true ore of mercury, the latter to the artificial compound. When formed in the cold, or in the wet way, the black sulphide is produced. This may be sublimed without decomposition when air is excluded, but when condensed the red variety, vermilion, is produced. Sulphur and mercury combine directly when triturated together, forming the black sulphide, these two elements having considerable affinity for each other. It is also produced by the action of alkaline polysulphides on metallic mercury. Sulphuretted hydrogen or alkaline sulphides precipitate it from solutions of mercuric salts. It is not attacked by nitric acid, nor by hydrochloric acid except in the presence of an oxidising agent, or of some metallic chlorides such as ferric or cupric chloride, when mercuric chloride is produced. Boiling concentrated sulphuric acid converts it into mercuric sulphate, sulphurous anhydride being given off:—

$$HgS + 4H_2SO_4 = HgSO_4 + 4SO_2 + 4H_2O.$$

The freshly precipitated black sulphide is partially decomposed by concentrated nitric acid with the separation of sulphur; it is also slightly soluble in caustic potash. Ordinary mercuric sulphide is, however, practically insoluble in either caustic alkalies or in ammonic sulphide.

When heated in the air it forms metallic mercury, and sulphurous anhydride is evolved. It is decomposed by heating with lime, alkaline carbonates, or metals such as iron, metallic mercury being liberated in each case.

Mercuric sulphide forms insoluble compounds with many other salts of mercury.

Salts of Mercury.—Both oxides of mercury form salts with acids, the salts so produced being characterised by a marked tendency to form basic salts containing excess of oxide of mercury, such, for instance, as $HgSO_4$, $2HgO$, and $Hg(NO_3)_2$, $2HgO$.

Mercurous Nitrate, $Hg_2(NO_3)_2$, 2 Aq, is formed by boiling excess of mercury with nitric acid diluted with one half its bulk of water, until crystals form. As a rule some mercuric nitrate is formed at the same time. It is soluble in a small quantity of warm water, a larger amount splitting it into an acid and a basic salt; several insoluble basic mercurous nitrates are known.

Mercuric Nitrate, $Hg(NO_3)_2$, nAq.—When mercury is dissolved in excess of nitric acid, and the solution evaporated, crystals of $2Hg(NO_3)_2$, Aq, are formed; crystals with varying proportions of water of crystallisation have also been produced. Mercuric nitrate is soluble in water, but it splits up readily into acid and basic nitrates; several insoluble basic mercuric nitrates are known to exist.

Mercurous Sulphate, Hg_2SO_4, is produced by heating excess of mercury with strong sulphuric acid or by rubbing up mercuric sulphate with metallic mercury; it is very slightly soluble in water, but dissolves in dilute nitric and in concentrated sulphuric acids.

Mercuric Sulphate, $HgSO_4$, is produced by heating mercury with excess of sulphuric acid and evaporating to dryness, when a white saline mass is left. It forms a white crystalline anhydrous powder, soluble in sulphuric

acid, but decomposed by water into a soluble acid and an insoluble basic sulphate.

Other Salts, such as chlorates, bromates, iodates, nitrites, phosphates, &c., are known, also carbonates which are very unstable. There are also a large series of mercurammonic compounds known, some of which are explosive; fulminating mercury belongs to this class.

The following table shows the characteristic reactions of mercurous and mercuric salts with various reagents :—

Reagent.	Mercurous Salts.	Mercuric Salts.
Sulphuretted Hydrogen or soluble Sulphides...	Black precipitate, insoluble in Alkaline Sulphides.	Black precipitate (at first white or yellow), insoluble in Alkaline Sulphides.
Potash or Soda ..	Black precipitate of Hg_2O, insoluble in excess.	Orange yellow precipitate of HgO, insoluble in excess.
Alkaline Carbonates	Dirty yellow precipitate, becoming black on boiling.	Yellowish red precipitate.
Ammonia & Ammonic Carbonate	Blackish grey ammoniacal compound.	White ammoniacal compound, soluble in great excess.
Soluble Chlorides or Hydrochloric Acid	White precipitate of Hg_2Cl_2 blackened by Ammonia.	No precipitate.
Soluble Iodides...	Green precipitate of Hg_2I_2 transformed by large excess into Hg and HgI_2, the latter dissolving.	Red precipitate of HgI_2, soluble in excess, also in large excess of Mercuric Salts.
Sodic Phosphate	White precipitate.	White precipitate (except with Mercuric Chloride).
Soluble Oxalates	White precipitate.	White precipitate.
Potassic Ferrocyanide	White precipitate.	White precipitate.
Potassic Ferricyanide............	Reddish brown precipitate.	White precipitate (except with Mercuric Chloride).
Soluble Chromates	Red precipitate.	Yellowish red precipitate.
Soluble Gallates	Brownish yellow precipitate.	Orange precipitate (except with Mercuric Chloride).
Potassic Cyanide	White precipitate of Hg_2Cy_2, splitting at once into Hg and $HgCy_2$.	White precipitate of $HgCy_2$, soluble in excess (except with Mercuric Chloride).

Physiological Effects.—All soluble salts of mercury are violent poisons when taken internally. The symptoms of acute mercurial poisoning are an acrid metallic and astringent taste; continual expectoration; severe pains in the stomach and intestines accompanied by vomiting and

diarrhœa; slow irregular pulse; severe syncope or sometimes convulsions, upon either of which death may follow. The best antidote in cases of mercurial poisoning is the white of raw eggs (albumen), which forms an insoluble compound with mercurial salts. Freshly precipitated ferrous sulphide is also a good remedy, forming insoluble sulphide of mercury and ferrous chloride which is harmless; it must, however, be promptly administered. Very fine iron filings may also be given immediately, as they will precipitate metallic mercury.

Slow poisoning is produced by exposure to mercurial vapours, or by protracted handling of mercury; its symptoms are salivation, skin affections, ulcers of the mucous membrane, and mercurial tremors. There is always liability to salivation when amalgam is being retorted, unless the operator takes especial care not to expose himself to the mercurial fumes. In these cases the mercury is slowly eliminated from the system. A strong solution of chlorate of potash is recommended as a mouth wash immediately after exposure to mercurial vapours.

Purification.—Mercury is generally bought and sold in flasks made of wrought iron closed with a screwed plug about three-quarters of an inch in diameter. These flasks hold 75 lbs. Spanish, equal to 76 lbs. avoirdupois, of mercury in Europe, and 76½ lbs. in California, and themselves weigh about 12 to 14 lbs. Commercial mercury is never quite pure; it is often contaminated with traces of foreign metals, those most frequently occurring being the more volatile ones, such as lead and zinc. Redistillation will not effect a complete separation from them, as some portion will distil over with the mercury; in order to avoid this, it has been recommended to distil at a low

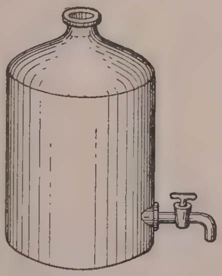
temperature in a current of superheated steam. It is, however, better to employ a chemical method of purification. When small quantities of lead or zinc are present, mercury distils much more slowly than when pure; other metals do not seem to exercise this action. Small quantities of mercury may be fairly well cleansed by vigorous shaking in a bottle, not more than a quarter full, with a little crushed loaf sugar, and afterwards filtering through a cone of stout blotting paper which has a small pin-hole at the apex of the cone. This method of filtration may always be used to remove suspended impurities.

It may be purified by distilling in a capacious retort, such as is used for retorting amalgam (see page 446), under a layer of cinnabar, the sulphur of which combines with the other metals present, and prevents their passing over. If fairly pure, it may be distilled under a layer of quicklime or iron filings, which will retain any sulphur or arsenic that may be present, and also prevent splashing. The retort should never be more than half full of mercury, and the distillation be conducted very slowly.

If it contains zinc or tin, it may be purified by digestion with hydrochloric acid or ferric chloride. Lothar Mayer recommends letting it trickle in a very thin stream through a column of ferric chloride about 4 feet in height. Brühl agitates repeatedly with a solution of potassic chlorate and sulphuric acid.

It may also be agitated with moderately concentrated sulphuric acid for some days. When heated with a solution of mercuric nitrate, the contaminating metals are dissolved as nitrates, mercury being precipitated. The most general method and the most satisfactory is by treatment with dilute nitric acid, in which it should be digested for twenty-four hours. The best plan is to

agitate with dilute nitric acid (about 1 of acid to 3 of water) in a tubulated glass or stoneware receiver (Fig. 1) furnished with a stopcock at the bottom, and capable of holding about a flask of mercury at a time. After thorough agitation it should be left at rest for a few days, and the pure dry mercury then drawn off as required from the bottom of the receiver by means of the stopcock. If crystalline crusts of nitrate of mercury form on the surface, more water, rendered acid by a few drops of nitric acid, must be poured on to it; the solution will not need renewing for a long time.



Scale ¾″ to 1 ft.
FIG. 1.

Due care must be taken that the stopcock is a strong one and well secured, as the column of mercury in the receiver exercises a heavy pressure upon it.

By keeping two such receivers in use, the mercury from one of which is being used whilst the other is allowed to purify slowly, being stirred up every few days, a supply of pure mercury may be ensured. It may be remarked that this method does not remove any of the precious metals, gold and silver, that may be present.

Pure mercury should form small, perfectly spherical globules, uniformly bright, which unite at once when brought into contact with each other, should leave no "tail" at all when allowed to run slowly down an inclined glass plate, should leave no film on rough blotting paper, and give no black powder when shaken up in a bottle with dry air.

CHAPTER IV

ALLOYS AND AMALGAMS OF GOLD

Gold is capable of combining with most of the better known metals, forming alloys. An alloy may be defined generally as a compound of two or more metals; that is to say, it is not merely a mechanical mixture of these metals, capable of being separated into its ingredients by mechanical or physical means alone, but these ingredients are united by so firm a bond as to cause the alloy to partake of the nature of a chemical compound; it differs from a true chemical compound, inasmuch as metals that are capable of forming alloys together, can do so in all proportions irrespective of their chemical combining weights. At the same time there are not wanting indications that more complete, intimate and homogeneous compounds are produced when the metals are combined in proportions according to their equivalents, and therefore capable of being represented by chemical formulas. The evidence that chemical compounds, albeit feeble ones, are formed, is moreover very strong. Thus, when metals are alloyed, chemicothermal changes are produced, the combination being mostly attended with the evolution of heat, but sometimes with its absorption. The physical

properties of an alloy are rarely, if ever, the mean of those of its components. Thus at times two malleable metals will produce a brittle alloy, or an alloy may be fusible at a lower or a higher temperature than any of its constituents. The colour of an alloy often differs considerably from those of the metals composing it; thus the red metal copper and the white metal antimony combine to form a violet-coloured alloy. The specific gravity of an alloy is rarely the arithmetical mean of those of its constituents. The chemical properties of the constituents become greatly changed when they are alloyed. The electric and thermic conductivity of an alloy often varies greatly from those of the metals contained in it. In fact all the phenomena which we are accustomed to associate with chemical combination are produced to some extent when an alloy is formed. An alloy of metals in the molten state may accordingly be looked upon as: (1) a chemical compound, or a series of chemical compounds, of the constituent metals; (2) a solution of one or more of these compounds in excess of any of its constituents; (3) a solution of one or more metals in another, provided that such metals may then be in an allotropic condition. When the molten alloy is allowed to cool, the solidified mass may still retain its ingredients in the above form, the dissolved bodies not separating out from their solvent on solidification. At times, however, we see that alloys, which are capable of existing at high temperatures, are decomposed on cooling. The real nature of alloys is at present the subject of investigation by numerous metallurgists, and microscopy has recently been found of especial value for revealing their true structure. The opinion most generally held seems to be that an alloy is a solidified solution of a metal or metals in

their "eutectic" compound, an eutectic alloy being one in which the constituents exist in such proportions as to produce the lowest possible freezing (or melting) point.

Alloys of gold have not yet been very carefully or very exhaustively studied. A great deal of our knowledge at the present day is derived from the labours of C. Hatchett, who, in 1803, studied the effect of various metals upon standard gold. He found, as might be expected, that most, if not all, metals tend to reduce the malleability and ductility of pure gold, the order in which they do so being as follows, the first on the list having the greatest effect in destroying these qualities:—

1. Bismuth.	Nearly equal	8. Nickel.
2. Lead.	in their	9. Tin.
3. Antimony.	effects.	10. Iron.
4. Arsenic.		11. Platinum.
5. Zinc.		12. Copper.
6. Cobalt.		13. Silver.
7. Manganese.		

This subject has received attention from Roberts-Austen,[1] who has investigated the effect of small quantities of various impurities (about 0·2 per cent.) upon the tenacity of gold. His results are reproduced in the table on page 69.

Pure gold was found to have a tensile strength of 7 tons and an elongation of 30·8 per cent. From all his observations he deduces the interesting law that the metals which render gold brittle are those which occupy high positions on Lothar Mayer's curve of the elements, that is to say, which possess a high atomic volume. $\left(\text{Atomic volume} = \frac{\text{Atomic weight}}{\text{Specific gravity}}\right)$.

[1] *Phil. Trans. of the Royal Soc. of London*, 1888, p. 339.

Element Present.	Tensile Strength per Square Inch.	Elongation per cent. on 3-inch Test Piece.	Percentage of Impurity Present.
	Tons.		
Potassium	0·5	Not perceptible.	Less than 0·2
Bismuth	0·5	,,	0·210
Tellurium	3·88	,,	0·186
Lead	4·17	4·9	0·240
Thallium	6·21	8·6	0·193
Tin	6·21	12·3	0·196
Antimony	6· (about)	?	0·203
Cadmium ...	6·88	4·4	0 202
Silver ...	7·10	33·3	0·200
Palladium	7·10	32·6	0·205
Zinc	7·54	28·4	0·205
Rhodium	7·76	25·0	0·21 (about)
Manganese ...	7·99	29·7	0·207
Indium	7·99	26·5	0·290
Copper	8·22	43·5	0·193
Lithium ...	8·87	21·0	0·201
Aluminium ...	8·87	25·5	0·186
Zirconium ...	12·	—	0·2 (about)

He further found that standard gold, consisting of 916·7 parts of gold and 83·3 parts of copper per mil. has a tensile strength of 18 tons per square inch with an elongation of 34 per cent. before breaking; the addition of $\frac{1}{2000}$ part of lead reduces this breaking strain to 5·5 tons per square inch, and $\frac{1}{500}$ part of lead reduces it still further to 1·84 tons, whilst the elongation has become immeasurable.

The specific gravity of alloys of gold with other metals is very rarely, if ever, the calculated mean of their constituents, there being either contraction or expansion in the volume of the alloy compared with the volume of its elements. The following table shows the amount of such change, according to Hatchett, in alloys made by melting 11 parts of gold with 1 part of the respective metals :—

Metal.	Contraction.	Expansion.
	Per cent.	Per cent.
Tin	1·875	—
Antimony	1·254	—
Bismuth	1·151	—
Zinc	0·316	—
Cobalt	—	0·107
Silver.	—	0·375
Lead	—	0·530
Nickel	—	0·716
Iron	—	1·572
Copper	—	2·416

Hatchett further remarks that, generally speaking, those metals which produce brittleness in gold when alloyed with it, also cause contraction when these alloys are formed; and he points out that the result obtained by him with zinc is not to be depended on, owing to the great volatility of that metal. Although bismuth causes contraction, and lead expansion of the respective alloys in the above proportions, yet when only one-half of a grain of either is present in an ounce of the standard gold-copper alloy, both of these metals cause great expansion, this being in the case of bismuth 4·72, and in that of lead 5·71 per cent.

Matthiessen has done a great deal of valuable work on this subject;[1] he has particularly investigated the specific gravities of the alloys of gold with tin, lead, bismuth, and silver in varying proportions.

In the case of tin he finds that the specific gravity of the alloy containing tin 100 parts, gold 1 part, calculated from the mean of its constituents, is greater than the observed specific gravity in the proportion of 1·0007 to 1; that for alloys ranging between 30 and 12 parts

[1] *Phil Trans.* 1860, p. 177.

of tin to 1 of gold, the former is less than the latter in the proportion of about 0·99 to 1; when the composition is between 8 and 2 parts of tin to 1 of gold, the former is again somewhat greater, and for equal parts of both metals or for the alloy of 2 parts of gold to 1 of tin again a little less, than the latter. In the alloy of 6 of tin to 1 of gold the two almost agree. In the case of lead the observed specific gravity is always greater than that calculated from the mean of the constituents in about the proportion of 1 to 0·99.

In the case of bismuth, the alloy of 90 parts of bismuth to 1 of gold had a specific gravity very slightly less than that calculated for it, whilst all the other alloys gave a slightly greater result, the difference increasing as the proportion of gold increased till the composition BiAu was reached.

He remarks that the alloys with lead and tin are all very brittle except where these metals were in great excess.

The following table represents the results of Matthiessen's experiments on the specific gravity of the alloys of silver and gold:—

Composition of Alloy.		Observed Specific Gravity.	Calculated Specific Gravity.	Ratio of Calculated Specific Gravity / Observed Specific Gravity.
Silver.	Gold.			
6	1	11·760	11·715	0·9961
4	1	12·257	12·215	0·9965
2	1	13·432	13·383	0·9963
1	1	14·870	14·847	0·9984
1	2	16·354	16·315	0·9976
1	4	17·540	17·493	0·9973
1	6	18·041	17·998	0·9976

His general deduction from all his experiments is that the contraction or expansion is a maximum when the metals are present in about equal parts. He has not, however, succeeded in developing any definite law on the subject.

Alloys of Gold with Various Metals

Gold and Silver.—The density of the resultant alloy is always very nearly the mean of those of its constituents; some observers make it a little higher, others a little lower. The results obtained by Matthiessen are given above. Whenever the two metals are combined in atomic proportions, they seem to form a perfectly homogeneous alloy if well incorporated. Thus, alloys having the formulas Au_2Ag, $AuAg$, $AuAg_2$, $AuAg_4$, $AuAg_{20}$ are quite homogeneous on cooling.

It is worthy of note that the alloy corresponding to the formula $AuAg_2$ is of nearly the same specific gravity as is mercury at ordinary temperatures; all those with more gold than 48 per cent. will accordingly be heavier, and all those with less gold lighter, than an equal bulk of mercury.

Generally speaking, the alloys of silver with gold are harder, paler, and more elastic than pure gold, the hardest consisting of 2 parts of silver to 1 of gold. The alloy of 70 parts of gold with 30 of silver is of a greenish tinge; it is used by jewellers, and known as "green gold." With equal parts of gold and silver, the alloy is white with barely a tinge of yellow, and with less gold than this, perfectly white. When the alloy contains less than two-thirds of silver, this metal cannot be completely extracted from it by nitric or sulphuric acid, and alloys rich in gold are not attacked at all. The

phenomenon known as "spitting," due to the absorption of oxygen by molten silver and the evolution again of the gas on cooling, is shown by alloys of gold and silver provided that the proportion of gold does not exceed one-fourth; it even takes place, to some extent, until the proportion of gold reaches one-third. If molten gold be poured into an equal weight of molten silver, the alloy can no longer retain the oxygen which the latter metal had absorbed, and it is given off with violent effervescence. Similarly, if the gold be melted and the silver then added in the solid state and allowed to melt without stirring, a layer of molten silver will form above the gold, and this layer will absorb oxygen. On stirring so as to incorporate the metals, an alloy is formed, and the oxygen thus liberated similarly escapes with much violence.

This possible source of loss must accordingly be guarded against when making gold-silver alloys, which is easily done by throwing a few pieces of charcoal upon the surface of the melting metal. The alloy containing 20 per cent. of silver is often spoken of as "electrum," this being the name applied to it by Pliny. It is derived from the Greek word for amber, and probably refers to the pale colour of the alloy as compared with pure gold.

Gold and Copper.—This is an important alloy commercially, forming the gold standard of most of the coinage of the world. British standard gold is twenty-two carats fine, that is to say, contains $\frac{22}{24}$ of gold or 916·6 parts per mil. and $\frac{2}{24}$ or 83·3 per mil. of copper. Its specific gravity is 17·157; considerable expansion takes place when this alloy is formed, the specific gravity calculated from that of its constituents being 18·47. The specific gravity of standard gold containing a quarter of a grain of lead to the ounce is 17·039, with half a grain

16·627, with four grains 17·032, and with eight grains 17·312. The melting-point of standard gold is 946° C. The French standard for coins is 900 parts of gold per mil., and for jewellery 750. The same coinage standard has been adopted by the other nations of the Latin Union and by America. The standard of the Transvaal was 916·9 per mil. If the copper be perfectly pure, it only lowers the malleability of the gold slightly, but even traces of impurity have, as previously indicated (page 68), a most important effect in this respect. The alloys with copper are harder, more fusible, and somewhat deeper coloured than pure gold. The hardest alloy consists of 7 parts of gold to 1 of copper. When heated in the air, these alloys rapidly become coated with a crust of oxide of copper. Alloys rich in gold are not affected by exposure to air, under ordinary conditions, but poor ones soon tarnish. Alloys having the chemical formulas Au_8Cu, Au_6Cu, Au_4Cu, Au_2Cu, are quite homogeneous when melted and well stirred, but $AuCu$ and $AuCu_5$ require to be well stirred, cast into ingots, and these ingots remelted with thorough stirring several times in order to obtain a homogeneous alloy. As in the corresponding case of silver alloys, copper can be dissolved out from the alloy when present in large proportion; when the alloy contains over 93·5 per cent. of gold it is not attacked by nitric or sulphuric acid.

Triple alloys of gold, copper, and silver are much used in the manufacture of jewellery, the silver being employed in order to make the colour of low standard alloys paler, and therefore more like gold, than would be the case if copper alone were used. Some of the very low standard alloys also contain a few per cents. of zinc, which is added with the same object.

Gold and Tin.—These metals alloy readily; a considerable percentage of tin makes gold brittle, but less than 2 per cent. does not seem to do it any harm. Alloys whose composition ranges between the formulas $SnAu_4$ and Sn_3Au_2 are vitreous and not crystalline in fracture. Sn_2Au (45·9 per cent. of gold) shows some tendency to form crystals, and has a granular fracture. All these alloys are of a yellowish gray to a grayish white colour. The alloys ranging between Au_2Sn_5 and $AuSn_6$ (40 to 22

Composition of Alloy.		Specific Gravity.
Gold.	Tin.	
4	1	16·367
2	1	14·243
1	1	11·833
2	3	10·794
1	2	10·168
2	5	9·715
1	3	9·405
1	4	8·931
1	6	8·470
1	9	8·118
1	15	7·801
1	50	7·441

per cent. of gold) crystallise in square prisms; the alloy Au_2Sn_5 has this tendency in a very marked degree, forming crystals consisting of secondary and primary prisms and pyramids of the tetragonal system, and having a well-marked basal cleavage. Their colour is tin-white, but on exposure to the air they tarnish to a bronze colour; these crystals "cry" like tin when they are bent.

The alloy containing 11 parts of gold to 1 of tin is pale

yellowish white, brittle, and has an earthy fracture of a yellowish gray colour.

Five per cent. of gold alloyed with tin scarcely affects either the hardness or the ductility of the latter metal. The table on page 75 shows the specific gravities of a large number of tin-gold alloys.

The relation of the actual to the calculated specific gravities of these alloys has already been indicated.

Standard gold which has one-half of its copper replaced by tin is brittle, pale yellow, and has a close-grained earthy fracture. With eight grains of tin per ounce it is fairly ductile, and rather paler than standard gold. It is, however, brittle at a low red heat.

Gold and Arsenic.—These metals are not easily alloyed by fusion in an open crucible, as most of the arsenic then volatilises; thus 5300 grains of gold melted with 450 grains of arsenic only retained six grains of the latter. This alloy is brittle, but bends slightly under the hammer before breaking. When a plate of gold at a red heat is exposed to the vapour of arsenic, a highly fusible, brittle, gray alloy forms, which melts and trickles off the surface of the plate. An ounce of gold melted in an atmosphere of arsenic absorbed 1·5 grains, forming a coarse-grained, gray, brittle alloy. When combination between the metals has taken place, it is very difficult to drive off the arsenic by heating the alloy unless the heat be intense and very prolonged. In one experiment, where heating was continued for an hour, six ounces containing 9½ dwt. of arsenic lost the whole of the arsenic, and also two grains of gold, which were apparently carried off in the vapour of the arsenic.

0·05 per cent. of arsenic will make gold unworkable under the hammer.

Gold and Antimony.—These metals alloy readily, although a little antimony is usually volatilised during the operation. The alloy containing 8 to 10 per cent. of antimony is grayish white, very brittle, with a close-grained, dull gray fracture. An alloy with the formula AuSb (61·5 per cent. of gold) is white and very brittle; it is superficially attacked by nitric acid. In all alloys of gold with antimony the latter oxidises on heating in the air.

Standard gold containing eight grains of antimony to the ounce forms a dull gray, brittle alloy; with half a grain it is brittle and close-grained, but shows a little metallic lustre; with a quarter of a grain it is still brittle, but not excessively so. Thus it would seem that 0·05 per cent. of antimony makes gold unworkable. Gold heated in the vapour of antimony, either in an open or a closed crucible, will absorb enough to make it very brittle and to change its colour to gray.

Gold and Bismuth.—These alloys are all brittle in any ordinary proportions. The alloy of 11 parts of gold and 1 of bismuth is greenish yellow, very brittle, with a fine-grained, earthy fracture. Molten gold readily absorbs bismuth vapour and is rendered brittle by it.

Standard gold containing eight grains of bismuth per ounce is pale brownish yellow and fine-grained; with four grains its fracture is coarse, with one grain coarse and spongy, with half a grain very spongy, with a quarter of a grain close granular; all these alloys are brittle. The table on page 78 shows the specific gravities of a number of gold-bismuth alloys.

In all these alloys, as already stated, there is strong contraction when the alloy is formed.

It may be noted that the alloy of gold and bismuth cupels readily when heated in the air; that is to say, that the molten alloy yields a readily fusible oxide of bismuth that will sink into and be absorbed by a porous body like a bone-ash cupel, leaving pure gold behind.

Composition of Alloy.		Specific Gravity.
Gold.	Bismuth.	
2	1	14·844
1	1	13·403
1	2	12 067
1	4	11·025
1	8	10·452
1	20	10 076
1	40	9 942
1	90	9 872

Gold and Lead.—All these alloys are brittle in spite of the malleability of the constituent metals; 11 parts of gold to 1 of lead yield a pale yellow alloy having a pale brown, earthy, fine fracture; with more lead the alloy becomes of a pale gray colour with a gray, fine granular fracture. Alloys with lead remain molten until they have cooled down to a temperature considerably below their melting-point, when they solidify instantaneously, emitting a bright flash, whilst the temperature at once rises to the melting-point of the alloy. Lead vapour combines freely with molten gold in closed vessels. Acetic acid dissolves out nearly all the lead from these alloys.

The following table gives the specific gravity of a series of gold-lead alloys:—

Composition of Alloy.		Specific Gravity.
Gold.	Lead.	
4	1	17·013
2	1	15·603
1	1	14·466
1	2	13·306
1	3	12·737
1	4	12·445
1	5	12·274
1	10	11·841

Standard gold containing nineteen grains of lead to the ounce has about the colour of standard gold, with an earthy fracture; with eight grains of lead the fracture is rather coarse-grained, with one grain coarse grained, with half a grain very spongy, with a quarter of a grain close granular. It is noticeable that both lead and bismuth affect standard gold similarly, half a grain of either metal to the ounce of gold yielding a very spongy alloy. The parallelism of their action is well shown in the following table of the specific gravities of standard gold when alloyed with varying amounts of the respective metals :—

Grains of Impurity per Ounce.	Specific Gravity.	
	Lead.	Bismuth.
38·0	18·080	18·038
19·0	17·765	not examined
8·0	17·312	17·303
4·0	17·032	16·846
0·5	16·627	16·780
0·25	17·039	17·095

The specific gravity of the standard gold was 17·157, of the lead 11·352, and of the bismuth 9·822. (Hatchett.)

It is further noteworthy, in this connection, that the alloy with lead, like that with bismuth, is capable of being cupelled, these two being the only ones that possess this property.

Gold and Zinc.—If gold and zinc be melted together in open vessels, much of the latter metal is volatilised, but in closed vessels there is but little loss. When gold is melted with brass very little of the zinc is driven off. Molten gold readily absorbs zinc vapours. It is stated that an alloy of 7 parts of zinc and 1 of gold can be completely volatilised at high furnace heats. Mr. Picard states that if a mixture of finely divided gold and zinc (as obtained by precipitating cyanidation liquors) be distilled in an ordinary retort, great loss of gold occurs, but that this loss can be reduced to about 0·05 per cent. by making the above mixture up into balls with molasses or strong sugar solution, so as to form a carbonaceous mass in the retort.

An alloy of 60 parts of gold with 1 of zinc is very brittle; 17 parts of gold to 1 of zinc give a pale greenish yellow alloy; equal parts yield a white, very hard metal, capable of taking a high polish; 2 parts of zinc to 1 of gold yield a fine-grained alloy, whiter than zinc; all these alloys are very brittle; those with more zinc are ductile. I have found that the alloy of 3 parts of zinc to 1 of gold is grayish white, granular and hard, but can be readily rolled into thin strips without annealing.

Standard gold containing nineteen grains of zinc to the ounce is pale yellow, has a coarse fracture, and is very brittle; with a smaller proportion of zinc it shows a little malleability.

Gold and Cadmium.—These metals alloy freely and

form gray and brittle alloys, although cadmium is a very malleable metal.

Gold and Iron.—These metals alloy readily; with 11 parts of gold to 1 of iron, the product is pale yellow and ductile, having a specific gravity of 16·885. If cast iron or steel be substituted for pure iron similar results are obtained. So-called "gray gold" consists of 4 to 5 parts of gold to 1 of iron; it has a grayish yellow colour, and is sometimes used in jewellery. Equal parts of gold and iron make a gray alloy, 1 of gold to 4 of iron a silver-white one, hard, magnetic, and capable of being tempered.

All these alloys are harder than gold; 4 per cent. of gold will make steel brittle. Standard gold containing nineteen grains of iron to the ounce is pale grayish yellow and ductile.

Gold and Cobalt.—These metals form alloys; that containing 11 parts of gold to 1 of cobalt is a dull yellow metal, with a pale earthy fracture, brittle, having a specific gravity of 17·112. Standard gold containing more than four grains of cobalt to the ounce is brittle, but with this proportion it commences to show signs of ductility.

Gold and Nickel.—The alloy of 11 parts of gold to 1 of nickel is brass-coloured, brittle, coarse-grained, with an earthy fracture, having a specific gravity of 17·068. Other alloys are said to be ductile, yellowish white, hard, as magnetic as nickel, and susceptible of taking a good polish.

Standard gold containing nineteen grains of nickel to the ounce is brittle with a fine-grained fracture; with eight grains to the ounce only slightly brittle, and with four grains ductile, and of about the same colour as standard gold.

Gold and Manganese.—These metals are known to form

an alloy, but nothing is known of its properties. It is prepared by melting gold at a high temperature in a brasqued crucible with manganic dioxide and carbon. The alloy, which is supposed to contain one-eighth to one-ninth of manganese, is pale yellowish gray, has a high lustre, and is slightly malleable.

Gold and Aluminium.—These alloys have been examined by Roberts-Austen.[1] With less than 10 per cent. of aluminium the alloys are pale yellow, but with this amount brilliantly white; from this point onwards as the proportion of aluminium increases, pink flecks appear until the alloy of 22 parts of aluminium to 78 of gold is of a splendid purple colour, in which intensely ruby coloured crystals may be recognised. On still further increasing the proportion of aluminium the alloys lose all red tinge and pass to a gray colour. It is also noteworthy that the alloy containing 10 per cent. of aluminium melts at about 630°, whilst the purple alloy has a melting-point of 1070° C., or rather higher than that of gold itself; the alloys with more aluminium have again lower melting-points. A small percentage (say about 2) of aluminium seems to increase the tenacity of gold, but gold containing 15 to 30 per cent. of aluminium is too brittle to test, the tenacity increasing again slightly as the percentage of the aluminium is still further increased. All these facts seem to point to the existence of a definite purple-coloured chemical compound of gold and aluminium having the formula $AuAl_2$. It is completely decomposed by hydrochloric acid, aluminic chloride being formed, and gold left behind in a very spongy state.

Gold and Platinum.—The alloy of 2 parts of platinum

[1] *Proc. Roy. Soc.*, 1891, vol. xlix. p. 347.

to 1 of gold is brittle; with equal parts a ductile alloy, having about the colour of gold itself, is produced; with 2 parts of gold to 3 of platinum the alloy is gray. The alloy of 11 parts of gold to 1 of platinum is yellowish white and ductile. According to E. Matthey, when a spherical mass consisting of an alloy of gold and platinum is slowly cooled, the platinum is concentrated to some extent in the centre of the mass.

Gold and Palladium.—These metals alloy in all proportions. The alloy of equal parts is gray, as hard as iron, and less ductile than either of its constituents; its specific gravity is 11·079. That of 4 parts of gold to 1 of palladium is white, hard, and ductile.

Gold and Rhodium.—The alloy containing a quarter of a part of rhodium is of a golden colour, very ductile, and difficultly fusible; with one-sixth of rhodium the alloy is rather more fusible, but less so than pure gold.

Gold and Osmium or Iridium.—These metals do not appear to form alloys. Iridosmium separates out in black grains, and sinks to the bottom of the crucible in which gold containing it is melted.

Gold and Potassium alloy when heated together; the alloy is decomposed by water, caustic potash being formed and gold left. The alloy containing 10 per cent. of gold takes fire when thrown into water, leaving the gold behind in the form of a black powder.

Gold and Sodium also alloy readily when heated together out of contact of air; the properties of the alloy are similar to the last-named.

Gold and Tungsten.—The alloy of these metals is yellow, and very difficultly fusible.

Gold and Molybdenum.—The alloy of 2 parts of gold to 1 of molybdenum is a black, brittle substance.

G 2

Gold Amalgam

As is the case with other metals, the alloy of gold with mercury is known as its "amalgam." Mercury alloys with gold with great avidity, this being due not only to the marked affinity of the metals but to the fact that mercury is a metal in the molten state at ordinary temperatures. Molten lead, for instance, will alloy with and dissolve gold quite as easily as will liquid mercury, the physical condition of the metal having a marked influence on its chemical relations.

Pure annealed gold placed in contact with clean mercury at ordinary temperatures is at once amalgamated, and so is a plate of gold exposed to mercurial vapours; mercury and gold are capable of uniting in all proportions, but it would seem that the resulting substance is a mixture of one or more definite amalgams of gold with excess of free mercury, or free gold, as the case may be. Professor Eggleston[1] has recorded the results of numerous interesting experiments, in which he shows that gold that has been heavily pounded, and is therefore in the unannealed condition, that is to say in a state of tension, will not amalgamate, or only with the very greatest difficulty, and he points out that these conditions are liable to be realised in the stamp-mill.

When gold is immersed in an excess of mercury, a chemical combination of gold and mercury appears to be formed first; and this combination (or amalgam) dissolves subsequently in the excess of mercury until the latter is saturated, when the remainder of the gold amalgam is deposited on the bottom of the containing vessel. The

[1] *The Metallurgy of Silver, Gold, and Mercury in the United States*, 1890, vol. ii. p. 586.

process may be aptly compared to introducing an anhydrous salt, such as anhydrous cupric sulphate, into a quantity of water more than sufficient to hydrate it, but not sufficient to dissolve all the hydrated sulphate of copper produced. As in the analogous case just quoted, so in the case of the amalgam, if more mercury be added, the whole of the amalgam formed can be dissolved. If the quantity of mercury be insufficient to dissolve all the amalgam, then the dissolved portion can be separated from the undissolved residue by any mechanical operation, such as filtration. It must be noted that the mercury wets (that is to say obstinately adheres to) amalgam, so that (just as in the analogous case of cupric sulphate) pressure has to be resorted to in order to free the solid residue as far as possible from the solution. In practice this is usually accomplished by squeezing through chamois leather or some other suitable substance. The solubility of amalgam in mercury is variously estimated by different writers. Lazarus Ercker, one of the first writers on this subject, wrote, in 1672, that "a hundredweight of mercury will carry some two or three ounces of gold and silver." Henry says that mercury squeezed from gold amalgam carries from a trace to ten grains of gold to the pound. The variations seem to be due to the fact that there are a number of different compounds of gold and mercury, and that the solubilities of these in mercury may vary; for it must be remembered that gold is not directly soluble in mercury as gold, but that it is gold amalgam, a compound of gold with mercury, that is soluble in mercury. In some experiments on this subject I have found that a mixture of 1 part gold with 50 of mercury produced a definite crystalline amalgam containing about 40 per cent. of gold. This was soluble in

mercury to the extent of about 2·29 parts of amalgam per thousand (equal to 0·94 parts of gold per thousand) at a temperature of 15° C. The higher the temperature the greater the solubility of amalgam in mercury. (See page 442.)

Thus with bar gold (containing silver) I obtained the following results:—

Temperature.	Amalgam dissolved in the squeezed mercury. Parts per 1000.
18° C.	3·27
21° C.	3·74
51° C.	3·81
72° C.	3·88

These figures suffice to show that the solubility does increase with the temperature, but not to determine the ratio of the increment, as different samples were operated on in each case.

This is corroborated by Kasantzoff,[1] who gives the following table:—

Temperature.	Parts of gold soluble in 100 of mercury.
0° C.	0·011
20° C.	0·126
100° C.	0·650

This author further states that variations of pressure in squeezing do not affect the solvent powers of the

[1] *Bull. Soc. Chim.*, t. xxv. p. 20.

mercury; this would indeed be expected from *a priori* reasoning.

From the above figures it may fairly be assumed that the average solubility of gold in mercury is about ten grains to the pound, and it is interesting to note that this would come to nearly 1·6 ounces to the flask, or just about as much as the value of the mercury itself.

Various more or less definite amalgams of gold have been isolated, and their characters described.

As already stated, native amalgam has the formula Hg_2Au_3, and contains from 39·02 to 41·63 per cent. of gold.

By treating 1 part of gold with 50 of boiling mercury, keeping heated for some days, allowing to cool and carefully squeezing, I obtained a hard alloy of gold and mercury, crystallising in silver-white, long, delicate, interlacing needles, and consisting of 41·43 of gold and 58·57 of mercury, thus corresponding exactly to the composition of native amalgam, but having apparently a different crystalline form.

The following list gives the results obtained by various experimenters on this point:—

$AuHg_4$ is obtained in hard white crystalline lamellæ, when precipitated gold is dissolved in mercury at 120° C. and the mixture allowed to cool.

Four-sided crystals of a yellowish white colour, easily fusible without decomposition, are said to have been produced, containing $AuHg_6$.

When a mixture of gold and mercury is heated carefully to a temperature a little above the boiling-point of mercury, till the weight of the residue remains constant, an amalgam is said to remain, having the composition Au_9Hg.

When gold is dissolved in mercury in about the proportion of 1 grain in 1000, and the mercury dissolved in dilute nitric acid at a gentle heat, an amalgam is left in four-sided prisms of a brilliant metallic lustre, which are not affected by boiling nitric acid and do not tarnish in the air. When heated they do not fuse, but give off mercury, and leave pure gold which retains the form and lustre of the original crystals. This amalgam contains about 88 parts of gold and 12 of mercury per cent., corresponding nearly to the formula of Au_8Hg.

Knaffl,[1] however, states that on long-continued heating of an amalgam of 20 parts of mercury to 1 of gold, and then acting on the mass with nitric acid, he obtained crystals of gold that obstinately retained small but variable amounts of mercury, which could, however, be removed by heating. The amount of mercury retained in the crystals is not definitely stated, and the experiment appears to be somewhat inconclusive.

By amalgamating gold and squeezing, a crystalline amalgam having the formula $AuHg_2$ has been produced. It will thus be seen that amalgams of more or less definite composition and form, having formulas respectively corresponding to Au_9Hg, Au_8Hg, Au_3Hg_2, $AuHg_2$, $AuHg_4$, and $AuHg_6$ have been isolated by different observers.

It seems that all these are decomposed into metallic gold and mercury at a heat somewhat above the volatilising point of mercury, but that, whereas those containing the larger proportions of mercury are fusible below this point, those richer in gold are not.

I have found in a series of recent experiments that if

[1] Dingler's *Polytechnisches Journal*, vol. clxviii. p. 282.

a soft amalgam of gold be kept heated to the boiling-point of mercury in a hard-glass tube, mercury will volatilise until an amalgam is obtained which is perfectly liquid at this temperature, but which sets hard, forming a silver-white mass, rather brittle, with a granular fracture, when cold; a good deal of heat is evolved at the moment of solidification, sufficient to volatilise some of the mercury, which forms a mirror round the mass of amalgam, the surface of which is at times left with a slight yellowish tinge. This elevation of temperature and consequent decomposition renders the determination of the exact composition of the hard amalgam a matter of some little difficulty; I find it to consist of: gold, 37·6 per cent.; mercury, 62·4 per cent.

The amalgam generally obtained in gold milling seems to have a composition corresponding to about $AuHg_2$, making allowance for the silver which it always contains, and this amalgam shows signs of incipient fusion at the temperature at which it commences to decompose. It is impossible to say, in the present state of our knowledge, whether it is a definite compound or a mixture of several of the series of alloys whose compositions are given above, perhaps also with free gold or free mercury as the case may be.

I have recently found that gold amalgam is attacked rather readily by solution of cyanide of potassium. Thus 18·76 grains of hard amalgam containing 37·6 per cent. of gold left for a week in a 1 per cent. solution of KCy, containing 16 grains of that salt, lost 0·179 grain of gold, no mercury at all being dissolved; in a similar 0·5 per cent. solution, 0·114 grain of gold was dissolved in the same time. This rate of solution is considerably greater than that of gold similarly treated.

It must be noted that when gold amalgam is decomposed by heating—gradually and at a low temperature at first, so as to avoid mechanical loss of amalgam by spirting or by the carrying off of solid particles of amalgam in the rapidly evolved stream of mercury vapour, but finishing nevertheless at a red heat—there is always some mercury obstinately retained by the residual gold, which can only be removed by actual fusion. This amount I find to be about 1 to 1½ per cent., when the heating is carried on properly. At the same time the mercury that is distilled off from the amalgam carries off in its vapour a certain amount of gold, which is truly volatilised. This is the well-known phenomenon of substances being vaporised at a temperature far below their volatilising point in a stream of the vapour of some other more volatile substance, and may conveniently be spoken of as "sympathetic volatilisation." The amount of substance thus volatilised depends upon numerous physical conditions, chiefly upon the temperature at which the operation is conducted, and also apparently upon the nature and shape of the containing vessel.

As the result of numerous experiments, I put the average amount of gold so volatilised in the ordinary operation of retorting amalgam at about 0·005 part per 1000. Of course when the mercury vapour is recondensed the gold vapour is condensed with it, so that mercury distilled off from gold amalgam contains normally the above amount of gold, or, say, about 1 grain of gold to 30 lbs. of distilled mercury.

CHAPTER V

PRIMITIVE METHODS—OUTLINE OF MODERN METHOD—WEIGHING ORE—BREAKING ORE—ROCK-BREAKERS—GRIZZLIES SORTING

Primitive Processes.—It is obvious that the most elementary process of extracting gold from quartz must consist of some simple method of pounding the rock and then washing the pounded stuff, until only free gold is left behind. This is still the basis of the ordinary method of prospecting a reef, and when a prospector comes across an exceptionally rich pocket containing much coarse gold, he still sometimes collects the latter by the rough-and-ready process of pounding the rich vein-stuff in a mortar, sifting out the larger fragments of gold, which, being malleable, will not be reduced to powder like the brittle quartz that accompanies it, and then washing the siftings in a prospecting pan so as to collect the fine gold, the entire process being sometimes spoken of as "hand mortaring." When somewhat larger quantities of ore have to be treated, a "dolly" is employed. This consists of a heavy piece of iron, often a partly worn out stamp-shoe, attached to a broken stem or to an iron bar suspended from the end of an

elastic sapling some twelve to fifteen feet long, the lower end of which is firmly driven into the ground (Fig. 2).

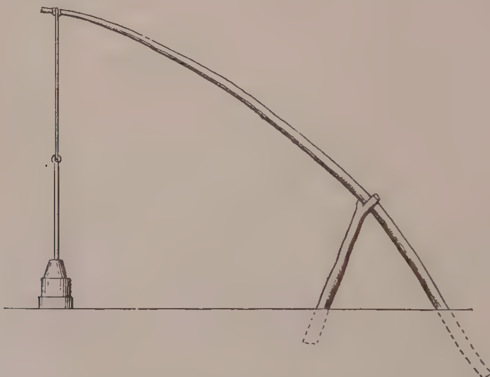


Fig. 2.

The weight is allowed to strike either on an old die or on something equivalent to it which forms an anvil, and which is surrounded with a box open at either end, made of boards nailed together, or else into a cast-iron mortar about eighteen inches high; of course, as this is always a rude make shift contrivance, built usually in some remote district, the details always vary according to circumstances and the materials available on the spot. A good dolly will crush about one cwt. of stone per day fine enough for panning purposes, say to a mesh of 0″·025 in diameter, or about thirty holes to the linear inch.

The most rudimentary method of quartz-crushing practised at the present day is probably that in use among the Ashantees and other tribes on the Gold Coast of West Africa. They break the gold-bearing quartz, which they manage to extract from the reefs, into small fragments between two stones, and then crush it down

to powder, using a fusiform piece of greenstone or granite, which is worked with a rocking motion forwards and backwards upon a smooth slab of stone, which is supported inclining gently away from the operator. The finely-ground stone is caught in flat wooden dishes one to two feet in diameter, their shape being that of a segment of a sphere. In these dishes the fine stone is skilfully washed in a pool of water, the dish receiving a peculiar circular combined with an undulating motion, by means of which the light quartz is washed over the edges of the dish and the gold retained. The tailings thus washed off are caught in a bigger dish, and rewashed several times until they yield no more gold. In some parts of China a very similar plan is employed, only in this case the crushing stone is very large, being worked by four men.

In some parts of the Andes a small two-stamp mill driven by water power, and resembling very much in general arrangement the old Saxon stamp mill, is used. It is, however, made without any iron whatever, being constructed entirely of wood, bound together with raw hide, the stamps being shod with granite boulders, and the mortar bottom being also a block of granite. The crushed ore is washed in a similar manner to that mentioned above, the wooden pan used for washing being, however, somewhat flatter. The latter is well known by its Spanish name of "batea," and is a useful implement for prospecting purposes.

The Chinese also pound gold quartz under tilt hammers, driven either by foot or by water power, and made entirely of wood except the hammer head, which consists of a boulder of quartzite. The pounded stone is washed in wooden dishes, the section of which is that of

a very obtuse cone, instead of being circular, as are those previously described. I have mentioned the above three crude native methods of crushing gold quartz because they are interesting, inasmuch as each apparatus is identical with (and perhaps derived from) one which is used in the preparation of some article of food in each of the respective countries. The negroes of West Africa grind maize, which is one of their staple food stuffs, between the two stones, whilst the South Americans and the Chinese use mills similar to those described for pulping coffee and husking padi respectively.

In the Middle Ages a mill was used on the continent of Europe which is practically the Saxon stamp mill as we know it. It was driven by a water-wheel, and had a wooden barrel from which wooden cams projected; the stamp stems were made of wood, and so was the coffer or battery box. The shoes were of stone or iron, and the ore was stamped dry or wet, the latter method having been introduced as an improvement on the former one about the beginning of the sixteenth century. An interesting wood-cut of one of these mills, taken from the well-known work of Georgius Agricola, *Vom Bergwerck*, published in 1557, forms the frontispiece of this volume. It gives a very good idea of the construction of a stamp mill three and a half centuries ago. Stamps essentially identical with these are still employed in some of the remoter gold-mining districts of Transylvania. The Saxon stamp mill of modern days and the Cornish tin stamp are practically still the same machine, with their wooden wearing portions largely replaced by iron. The weight of the stamps and the general efficiency of the machine have, of course, been increased, but the principle and many of the details remain unchanged. A series of alterations

of each essential portion, still retaining, however, the leading mechanical principle, has evolved the modern Californian gravitation stamp mill, the minute details of which as now constructed will be considered in the following chapters.

Modern Process.—The modern method of treating gold quartz has become a somewhat complex operation. The weight of the ore is usually determined either before or after delivery to the mill; it is next broken down to a proper size for treatment in the stamp mill, where milling operations proper commence, the ore being fed into the mill and there crushed to a suitable degree of fineness. The first step is the extraction from it of the free gold, and afterwards of the combined and "rusty" gold. The free gold is always caught by amalgamating it, and then collecting the amalgam, the great object being to secure the valuable metal at as early as possible a stage of the process. The operation of amalgamation may be carried out in the battery box itself, or outside it on amalgamating tables, in mercury wells, by special amalgamating apparatus (in pans, &c.), or by a combination of several of these methods. The heavier portions of the escaping pulp, which contain in many cases the largest portion of the remaining gold, are then separated by some method of concentration, and the concentrates so obtained are treated, to obtain from them the gold that they contain. Sometimes when there is comparatively little free gold this is not separated first, but the pulp is subjected to concentration directly it leaves the battery, and the concentrates, which then contain the free gold as well, are put through one or several processes to obtain the gold from them, the free gold being either collected separately, or else got together with

that held in the sulphurets. If the pulp still contains a notable proportion of gold, it is next run into settling pits, and the sand so collected is subjected to further treatment, now mostly by chemical methods; in some cases the preceding operation of concentration is omitted, and the whole of the pulp treated chemically as soon as it has left the amalgamating appliances, and in a few extreme cases amalgamation even is dispensed with. The best modern practice consists in amalgamating in the battery box, and then on copper tables, any escaping particles of amalgam being arrested in suitable amalgam traps. If there is then sufficient gold in the pulp to repay its further treatment, as is mostly the case, it is either treated direct by a chemical method such as cyanidation, or more often is sized and concentrated, and the concentrates are then treated specially by some metallurgical process that will extract the gold, the tailings being again further treated, if necessary, by some such process as cyanidation. This general scheme may not be universally applicable to all ores. I believe, however, that there are very few indeed that will not give better results by this system than by any other, provided only that scrupulous attention is paid to all its minute details, which must in every case be specially adapted to the ore under treatment. Very often a full knowledge of these minutiæ can only be obtained experimentally, but the proper guide is in every case a thorough study of the chemical and physical characteristics of the ore. It must be noted that there are numerous mills working, and working successfully, on principles at variance with the above, in different parts of the world, and that these local processes have become firmly established and do not seem likely to be given up very readily. It is quite

probable that the above method, which may now be looked upon as a standard one, namely, battery amalgamation, followed by copper tables, concentration, &c., would give with the ores in question at least as good and very possibly even better results; but as these special processes have been used with success and profit for many years, as the mills have been designed specially for them, as the men are well trained in their practice, and as the whole of the operations are thoroughly organised with special reference to them, it is not likely that these systems will now be changed, and it is even highly probable that were they changed for the modern method, the latter would not at the outset give better results, even if as good, whatever it might do ultimately. These facts must not, however, be construed into proof that such local methods are intrinsically better than the modern standard one. I do not propose in this volume to enter into details of any methods except the most modern, my object being rather to point out the principles underlying the practice of scientific gold extraction, than to enlarge upon variations in details thereof, which appear to me to possess only local, or at most historical interest.

General Arrangements.—The general arrangements of stamp-mills must vary very greatly according to the circumstances of each individual case. Thus a customs mill—that is to say, a mill that is run for the purpose of crushing ores belonging to outside proprietors, who simply pay for the use of the mill—must vary in its structure, as it varies in its objects, from a mill erected for the sole purpose of crushing the ore extracted from a particular mine, and of these, again, the small mill erected to treat the quartz from a small but very rich vein should differ

considerably from a large one whose object is to deal with enormous quantities of low-grade ores. And it is evident that unless the construction of a mill be strictly proportioned to the duty required of it, it can never prove a satisfactory machine.

Weighing Gold Quartz.—In the case of a customs mill the ore is usually delivered in carts or waggons, more rarely packed in bags or sacks. In the former case each vehicle should pass over a platform scales, there be carefully weighed, and the ore dumped into the particular bin assigned to it, the tare of each vehicle being also noted. If the ore is in sacks or bags, their tare, as well as their gross weight, must also be ascertained. When many different classes of ore have to be treated at a mill, the ores should always be weighed and not measured, as their specific gravities are liable to vary within very wide limits. The condition of dryness or wetness of the ore should also be recorded at the time of weighing. Disputes as to the weight of quartz delivered at the mill are sure to arise unless such precautions are observed; and to obviate them it is preferable, whenever possible, that the stone should be weighed in the presence of a representative of its owners.

When a mill is run in conjunction with a mine, the method of determining the weight of the stone by measurement is far preferable to that of weighing it; once for all, at the commencement of a campaign, a unit of measurement should be fixed upon, this being generally a car of the size used in conveying the ore from the mine to the mill. In the majority of instances these points are connected by a tramway, and a tramcar properly filled makes the best unit of measurement. Sometimes, on account of the physical features of the country, it is found

convenient to dump the ore as delivered from the mine into a hopper, and to use other cars to tram it thence to the mill; in this case the latter car will form the unit. It should be the duty of the man at the rock-breaker, when this machine is used, to see that the cars are properly filled, and that a proper tally is kept of the number of cars brought to the mill. When the weight of ore in the unit car is determined, this should be done by weighing a large number (say twenty to thirty) cars of ore on each of three or four successive days, the percentage of moisture in the ore being carefully determined each day on a large average sample. This is easily done by weighing out, say, 200 lbs. of well-mixed ore and drying it on an iron plate over a wood fire, stirring it with a piece of flat iron until no more moisture comes off, but never allowing the iron plate to become hot enough to char a chip of wood laid on it; in the case of pyritous ores special care must be taken not to allow the heat to rise high enough to decompose any of the constituents of the ore. The dried ore is reweighed carefully, and the amount of moisture thus determined. This figure will be found most useful in subsequent calculations. The percentage of moisture in gold ore varies within very wide limits according to circumstances; it may range from 3 to 15 per cent., or even more occasionally. It must not be forgotten that the specific gravity of ore from different parts of even the same reef is liable to vary considerably; thus quartz heavily charged with pyrites from a low level will be much heavier than the gozzany quartz from the upper levels, resulting from the decomposition of this same stone. Accordingly a fresh determination of all the data regarding it must be made when ore from new levels or new working faces is first brought to the mill; such

ores are for many reasons best kept separate throughout the process of milling, as they will probably require different methods of after-treatment.

Of course all that is really required to be known for the purposes of the mining engineer is the amount of gold obtained from a given unit of volume (say a cubic foot) of the stone, as it stands in the reef. This being known, the engineer could frame all his calculations by means of this one datum only. At the same time, however, it is advisable to base all calculations upon the ton of ore, so as to have a standard of comparison to enable the efficiency and economy of the mill in question to be compared with others in the same district, or in other parts of the world. It is, of course, the mine manager's business to know how much a cubic foot of stone as it stands in the reef weighs when extracted and sent to the mill. It is assumed that only milling ore is sent to the mill, all the worthless portions, mullock, casing, &c., being picked out before it is loaded into the mill car. In the rare cases, which may occur with very small mills having no rock-breakers, where it is found expedient to do this work either wholly or partly at the mill, a record must be kept of the stone so rejected.

The weight of ores varies within very wide limits, a spongy dry quartz containing little or no metallic sulphurets being obviously far lighter than a wet dense quartz heavily charged with galena, for instance. Broken quartz as sent to the mill will generally range between fifteen and twenty-two cubic feet to the ton. A convenient size of truck for handling and dumping is one carrying about half a ton of stone.

It may be noted that there is a growing tendency to measure gold quartz by the short ton of 2000 lbs. (equal

to 0·893 long ton), this practice obtaining in the United States, in Canada, and in the Transvaal. It is to be hoped that this practice will extend rapidly, and that the long ton of 2240 lbs. will soon become obsolete, together with such purely local methods of measurement as the Colorado cord (equivalent to about seven short tons of ordinary quartz).

Breaking the Quartz.—Ore as it comes from the mine is in lumps too large for feeding direct into the stamp-mill, which is rarely arranged to take a piece as large as three inches cube. Moreover, even if it could take such large pieces, it is by no means desirable that they should be fed into it, as breaking quartz in the mill is a costly process, and takes up time during which the mill might be more profitably employed in fine crushing. Feeding large lumps into the mill is also apt to break the screens and damage the mill in other ways, whilst uniform steady working would be an impossibility. On all these grounds, but principally on the score of economy, it is advisable to break the ore very small before feeding it into the mill, seeing that it costs far less to break down stone in the rock-breaker than under the stamps. As a general rule it should be broken to pass through a $\frac{3}{4}$ inch ring. It will occasionally, but rarely, happen that ore has to be broken by hand, in which case it cannot well be broken as small as above recommended. This will only occur in the case of a very small mill, when funds have perhaps not proved sufficient to provide a proper rock-breaker. Or, again, a small mill may be working on rich, narrow leaders of quartz in hard, barren country rock, where circumstances render it advisable to put the rich quartz alone, or at any rate as clean as possible, through the mill. It may then become necessary to break the stone

by hand as it comes from the mine, in order to be able to reject all the barren portions. The conditions under which hand-breaking may deserve the preference over machine-breaking can be summarised as follows:—

(*a*) A small mill treating a small output of rich ore.

(*b*) Soft ore, easily broken.

(*c*) High cost of motive power.

(*d*) Very cheap and fairly efficient unskilled labour, whilst skilled labour, such as that of engineers, mechanics, &c., commands high prices.

I have notably found these conditions combined in the Kochgar district of the Urals, where the ordinary labourers employed in breaking quartz get only 11*d*. per day, whilst skilled mechanics are scarcely to be got at all; all the fuel used has to be imported from a distance, and much of the ore treated is a soft, partially decomposed granite. Although a mill in this district had a rock-breaker erected, it was found cheaper not to run it, but to break the ore by hand. In some parts of West Africa a somewhat similar state of affairs prevails.

Whenever hand-breaking has to be resorted to, this should be done in a spacious shed, well lit and floored with earth rammed hard, or heavy plank covered with stout sheet iron, so as to admit of the floor being swept up from time to time, as the fine dust formed in breaking high-grade ore is always found to be very rich in gold. Sometimes the breaking is carried on in a portion of the mill building itself behind the stamp-mill. The breaking should be done on a bed of the rock itself, broken small; and a light stone-breaker's hammer, double egg-ended, about three pounds in weight, should be employed, the handle being about two feet six inches long and somewhat flexible. A heavy sledge-hammer, say twenty pounds in

weight will be required for the large lumps. Men engaged in this work mostly protect their legs against cuts from sharp splinters of quartz by means of a couple of folds of sacking fastened round their shins, and they sometimes also wear goggles. The latter precaution is unnecessary when the men are used to their work.

Rock-breakers.—There are two main types of these now being largely used—namely, the old type where the stone is crushed between a flat fixed jaw and a reciprocating one, and the newer type in which the fixed jaw is circular, whilst the other one gyrates inside it. In either case, the wearing portion of the jaws consists of dies which are capable of renewal.

To the first type belong the well-known Blake, the Marsden, Dodge, Foster, and numerous other crushers, which may be looked upon as merely variations of the first-named, the Gates and Comet being the best known circular ones. The conditions to be observed in a good rock-breaker are: that the machine must have sufficient weight to work steadily; that it must be strong enough to resist the very severe strains to which it is exposed, these strains tending, firstly, to burst open the ends, and, secondly, the sides, of the machine; that the jaw dies be capable of rapid and easy change, removal, and renewal, and that they be capable of being fixed firmly and adjusted accurately; that all wearing parts be accessible and capable of renewal; that all parts needing oiling, and above all the driving shaft, be kept as far away from the mouth of the machine as the size of the latter will admit of, in order to prevent the oil, which is used as a lubricant, from finding its way into the quartz; that it have a heavy fly-wheel or fly-wheels, and a fast and loose pulley with good sliding fork for the belt, in a position convenient for the man in charge.

One of the best materials for the dies is good chilled cast-iron, cast with corrugations about two and a half inches from point to point. Steel plate has been used, but wears very fast; cast steel, especially manganese or chrome steel, appears to give very satisfactory results. As a rule, the die plates are so constructed as to be capable of being inverted when the lower edge, which has most of the work to do, becomes worn out. The average wear of good chilled iron dies may be taken at 0·1 lb. of metal per ton of stone broken. There are several ways of securing the dies. The method of running them in at the back with lead or some similar readily fusible metal is not a good one, as the dies are then difficult of removal, and there is moreover a risk that particles of this metal may find their way into the quartz and subsequently into the bullion, which they would render brittle. The system of through bolts with heads counter-sunk on the working face of the dies is also not to be recommended. The best one probably is that in which the dies are secured by T-headed bolts entering into slots in the back. These dies should be cast of chilling metal of such quality as to secure a thoroughly hard face, whilst the back remains sufficiently soft to admit of chipping strips being planed to fit accurately similar planed strips that form a portion of the permanent jaw; or, if the chilling quality of the metal used is too hard for this purpose, it will do almost as well to cast strips of flat iron into the back of the die, which strips can then be planed as before. The space between the chipping strips should be filled with strips of dry deal in order to form a solid, slightly elastic cushion for the dies to rest against.

Blake Crusher.—This is too well known to need any detailed description. It consists of a heavy rectangular metal frame, the front portion of which forms the fixed

jaw. The swinging jaw is actuated by a toggle-joint at its lower end, the pitman being worked by a powerful eccentric forged on the driving shaft of the machine. Various devices are employed for adjusting the width of the aperture between the bottom ends of the jaw faces. A very good form of this crusher is the so-called Blake-Marsden rock-breaker, in which the driving shaft is set as far back as possible from the mouth, the swinging jaw being drawn back at each stroke by the action of the machine instead of by a spring.

Sectional crushers are manufactured for special cases where the transport of the heavy castings that compose the ordinary pattern is impossible; these machines are usually built of thick steel plates, the strain being taken by massive steel through bolts. A good pattern is manufactured by the Union Iron Works of San Francisco. Sectional crushers rarely do good work; in the first place, the total weight of the machine has usually to be kept so low that sufficient rigidity cannot be obtained, and part of the force that should be employed in breaking the stone is accordingly spent in racking the machine. Moreover, the severe strains to which it is subject usually end in causing more or less damage to those parts chiefly exposed to them, however strong these may be at the outset. Such crushers, and built-up crushers generally, should only be employed in case of absolute necessity; the heavy cast-iron frame is always to be preferred when it can be obtained.

The efficiency of this type of crusher is dependent on many circumstances, one of the chief items being of course the quality of the quartz. The following table, the data of which are only approximate, and subject to considerable variation according to circumstances, will

give some idea of the capacity of machines of good construction working on typical quartz, the fly-wheel shaft of the machine making 200 or 300 revolutions per minute :—

Dimensions of Mouth.	Tons of Quartz crushed per hr.	Weight of Machine.	I.H.P. required.
10″ × 4″	3	2½ tons	9
10″ × 7″	5	3½ ,,	12
15″ × 10″	7	5 ,,	18
20″ × 12″	9	7 ,,	25
24″ × 18″	12	9 ,,	30
30″ × 24″	16	12 ,,	45

Dodge Crusher.—This machine differs from the Blake mainly in that the point of suspension of the vibrating jaw is at the lower instead of at the upper end. The

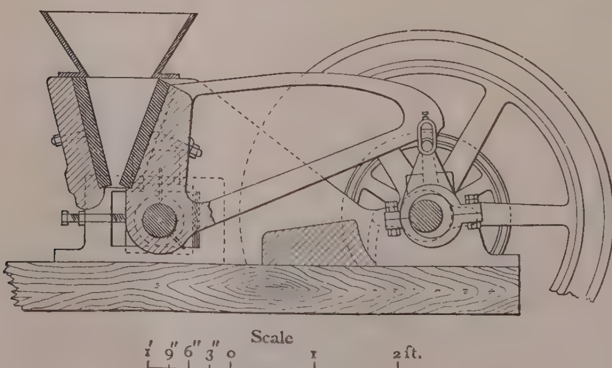


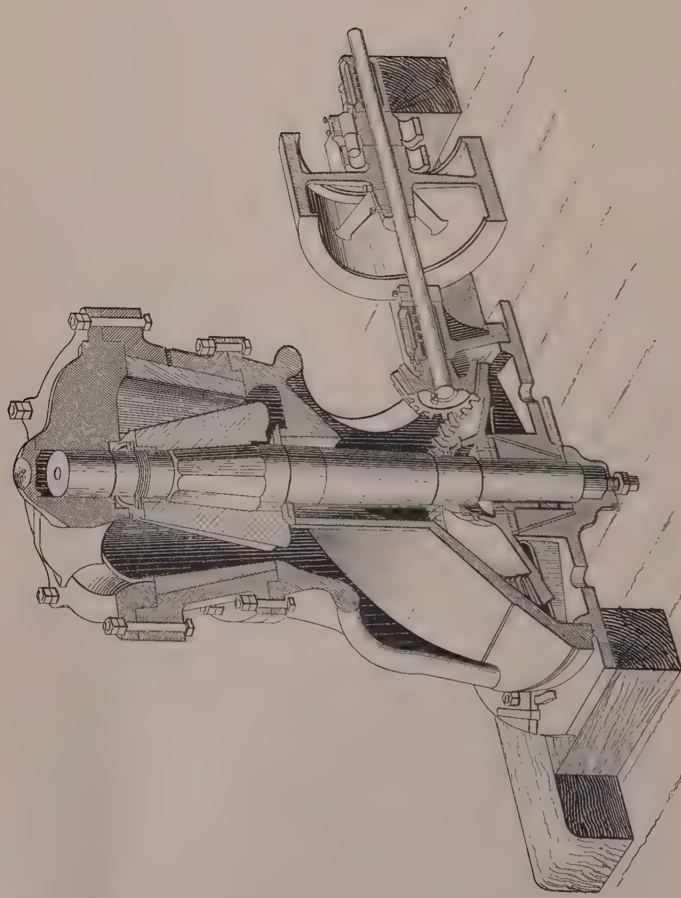
Fig. 3.

advantage of this arrangement is that it breaks the ore to a far more uniform size ; at the same time its capacity

is correspondingly diminished. This machine is shown in section in Fig. 3, from which it will be seen that the distance between the jaws can be readily adjusted by the screw that shifts the bearings of the swinging jaw inwards or outwards. The following table will give an idea of its average efficiency:—

Dimensions of Mouth.	Tons of Quartz crushed per hour.	Weight of Machine.	I.H.P. required.	Number of revolutions per minute.
9″ × 7″	1½	2 tons	8	235
12″ × 8″	3	3 ,,	12	220
16″ × 10″	6	5 ,,	20	200

Gates Crusher.—This is the best known of all the gyrating crushers. As will be seen from Fig. 4, it consists essentially of a cylindrical casting, within which fit a set of dies which together form a jaw of the shape of a truncated cone pointing downwards. This is partly closed by the gyrating head that forms the moving jaw, and which carries a die in the form of a cone pointing upwards. The quartz is crushed by its descent in the annular space between these two, this interspace being wedge-shaped in cross-section. The crushing head is carried on a shaft slightly inclined from the vertical, suspended at its upper end, the lower end being carried round in a small circle, the head being so arranged as not to revolve, but only to receive the gyratory motion of the suspended shaft. This type of crusher is capable of doing a great deal of work, and does not consume much power, while it runs more steadily and with less jar than the reciprocating crusher. The chief objections to it are its great weight and its slightly more



Section in Perspective

FIG. 4.

complicated construction. There is little to be gained by the employment of the smaller sizes of this machine, but it can be recommended where a large crusher is required, and above all where ore-breaking is done in a separate rock-breaker house, for which it is well suited. The following table exhibits its approximate capacity under normal circumstances:—

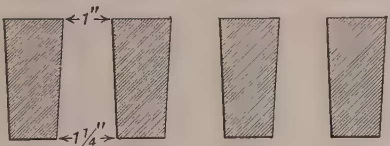
Diameter of Hopper.	Dimensions of each of the three openings at Mouth.	Weight of Machine.	Tons of Quartz crushed per hour.	I.H.P. required.	Number of revolutions per minute.
28″	10″ × 4″	$1\frac{1}{2}$ tons	$1\frac{1}{2}$	10	500
$37\frac{1}{2}$″	12″ × 5″	$2\frac{1}{2}$,,	$3\frac{1}{2}$	16	475
$39\frac{1}{2}$″	14″ × 6″	$3\frac{1}{2}$,,	5	20	450
$44\frac{1}{2}$″	15″ × 7″	6 ,,	8	30	425
51″	18″ × 8″	9 ,,	12	40	400
59″	20″ × 10″	12 ,,	20	50	375
66″	24″ × 11″	16 ,,	25	60	350
120″	30″ × 13″	27 ,,	35	80	350
132″	42″ × 18″	40 ,,	85	150	350

Grizzly.—An important adjunct to the rock-breaker is the grizzly. This consists of an inclined plane of iron bars leading down to the mouth of the rock-breaker, the bars being set at a distance apart equal to the space between the bottom ends of the rock-breaker jaws. Thus if the rock-breaker be set to crush to a $\frac{3}{4}$-inch cube, the grizzly bars must be $\frac{3}{4}$ inch apart. All the stone that is small enough to go direct to the mill will accordingly pass through the grizzly bars, and not go into the rock-breaker at all; the material passing through the bars drops into a shoot which delivers it into the same hopper into which the rock-breaker itself discharges. The grizzly is thus an arrangement for increasing the

efficiency of the rock-breaker by throwing less work upon it. It follows that when the rock-breaker's capacity is considerably superior to that of the mill the grizzly is but little needed, especially when the rock-breaker is run by water power. Even then, however, it will effect some slight saving in the wear and tear of the rock-breaker. This saving will vary in proportion to the size of the fragments which the rock-breaker is set to deliver. It is evident that if a rock-breaker were set to break to 2-inch cube, there might be, say, 20 per cent. of the entire product of the mine below this size, and a grizzly with bars 2 inches apart would thus only leave 80 per cent. of the mine product to be dealt with by the rock-breaker. Under the same conditions of mining, if the ore had to be delivered to the mill broken to $\frac{3}{4}$-inch cube, there might probably be only some 5 per cent. of the mine product below this size, and the saving of this small amount of extra work to the rock-breaker might not compensate for the extra expense and trouble of erecting and maintaining a grizzly. No general rule can, however, be laid down, and the nature of the ore as it comes from the mine should be carefully studied before the erection of a grizzly is decided on; moreover, it must not be forgotten that a grizzly needs a good deal of head-room.

Grizzlies are usually from 3 feet to 6 feet wide, and the length of the bars varies from 10 feet to 15 feet; these are usually set at an angle of 45° to 55°, according to the nature of the ore. The bars are made of either iron or steel, the latter being preferable, about 1 inch wide on the face and about 3 inches deep. It is advisable to have bars specially made for this work having a section as shown in Fig. 5, which represents a number

of the grizzly bars in section, the bars being $\frac{1}{4}$ inch wider on their upper than on their lower faces; this arrangement obviates any risk of their becoming choked. Old steel rails with the heads turned downwards make excellent grizzly bars. They are kept their proper width apart by means of distance pieces at either end, and by having bolts about an inch in diameter with washers on them to fit between the bars running through the entire system. The upper portion of the grizzly should consist of a plate of iron on to which the car is dumped. The grizzly sometimes discharges direct into the mouth of the ore-breaker; this should only be done when the grizzly is supplied from an ore bin having a door so regulated that just sufficient stone passes out to keep the crusher steadily at work, which at the same time necessitates a certain amount of uniformity in the size of the stone as it comes from the mine. When practicable this is an excellent arrangement for economising labour, as no one is required to attend to the rock-breaker except for such casual supervision as it will receive from the mill man. The more usual arrangement is to discharge the stone on to a rock-breaker floor set level with the top of the mouth of the machine, and covered with stout iron plates. On this floor stands the man in charge of the breakers. His tools consist of a stout iron hook or scraper to enable him to drag the ore into the mouth of the machine, one or two steel bars to turn over heavy lumps, and a square-mouthed steel



Scale 3" to 1 ft.

FIG. 5.

shovel for small stuff. One man at the breaker and one dumping cars and attending to the grizzly can handle as much stone as the biggest size rock-breaker can break in a ten hour-shift. As a rule the rock-breaker is run during the day shift only, except in case of emergency, when the mill is short of ore.

The best possible place for the rock-breaker, especially in the case of a large mill, is in a separate rock-breaker house situated either at the mine or between the mine and the mill, wherever sufficient head-room can be obtained for it. If, for instance, the ore is raised from a shaft, it is advisable to hoist to a height of, say, 20 feet above the level of the tram-line which supplies the mill, and to deliver the rough ore (after it has been picked over if necessary) direct to the shoot supplying the grizzly or rock-breaker as the case may be, whilst if the bottom adit is for any reason a similar height above the tram level, this head-room should similarly be made available. The best system is to deliver the ore to a large supply bin, whence it passes over the grizzlies and through the rock-breakers, and thence into storage bins below the latter, which bins should be made as large as circumstances admit of, and furnished with shoots and doors so as to automatically fill the mill cars.

A good rule for the capacities of the storage bins is that they should be able to contain at least a twenty-four hours' supply for the mill, so that the latter can run continuously, even if the mine is stopped for a day.

The system of locating the rock-breaker in a separate breaker house was first employed in California, the once famous Plymouth mine being a good example of it; in this mine the rock-breaker, run by a Pelton wheel, was

placed at such a height in the headgear of the mine-shaft as to enable cars to be run from beneath it along a raised tram-line which was laid above the mill bins. This method never seems, however, to have become very popular in the United States, but has reached a high stage of development in the large mills of the Witwatersrand. Here the so-called "crusher stations" are either placed close to the headgears of the winding shafts, or when a large mill is supplied from several such shafts a crusher station is best placed at some intermediate point; it may then best be constructed close to the mill, but still quite independent of it. Such a crusher station contains the grizzlies, rock-breakers, storage bins, and, in some cases, sorting floors, to which reference will be made subsequently. Quite recently the method of using two sets of rock-breakers, one above the other, as advocated for some time by the writer, has come into use in a few mines, the very expressive term of "tandem" crushing being now generally applied to it. In such a case the ore is first tipped on to a coarse grizzly, the coarse ore going direct to the first or coarse crusher, whilst the ore that goes through falls on to a fine grizzly; the fines from the latter go direct to the mill, and whatever does not pass through, to the fine crusher. The product from the coarse crusher also goes to a fine grizzly, which separates out the fines that are ready to be milled, whilst the larger pieces go to the fine crusher. Sometimes all the ore that has passed the fine crusher is also dropped on to a grizzly. and all pieces that are too coarse are returned to the breakers to be crushed finer. In a few cases tandem crushing is combined with tandem sorting, the ore undergoing a sorting between each crushing.

When head-room for a separate rock-breaker house is

not available at the mine, or when for any reasons it is not considered worth while building a separate rock-breaking station, the rock-breaker house has to form part of the general mill-building, a special rock-breaker floor being provided at the right level. This arrangement, which is still perhaps the most general, is illustrated in several of the views of stamp-mills, Figs. 67 to 71. When possible it is advisable to have one rock-breaker to every 20 heads of stamps, although it has been found quite feasible to supply 40 heads from one machine ; this usually entails, however, some little difficulty in duly distributing the crushed stone to the various battery hoppers. At the same time it must be distinctly remembered that crushing in the rock-breaker is performed at a far less cost (less than one-fifth as a general rule) than in the stamp-mill. It is therefore wise economy to crush as fine as possible in the rock-breaker, even if it be necessary to double the rock-breaker capacity of the mill, and in cases where very large lumps of ore have to be handled, it may, as in the case of crusher stations, be advantageous to pass these through two rock-breakers, the first to break, say, to a 3 inch cube, and the second to reduce this stuff down to $\frac{3}{4}$ inch. Double rock-breakers have been constructed for this purpose, consisting practically of two ordinary rock-breakers, one above the other, driven from one shaft, and with the sizes of their jaws properly proportioned for continuous work; these are, however, only suitable for very small mills. When two sets of rock-breakers, one above the other, are put in, the upper ones may with advantage be constructed on the gyratory principle, whilst for the lower ones the Dodge crushers are best.

It is, however, always very advisable to have the rock-

breakers in a separate building from the mill proper, because the rock-breaker is a machine that must necessarily cause much jar and vibration when working, and this defect is greatly accentuated when one or more of these are supported on a floor 30 feet or so above the floor of the stamp-mill. Again, during their operation much dust is produced, and this fine quartzose dust is apt to find its way into the bearings of the comparatively delicate pieces of machinery at work in the mill, to—it need hardly be said—their serious injury.

The separate crusher station no doubt adds to the first cost of the plant in most cases, and may also add somewhat to the working cost, especially where the only source of power is a high-priced fuel. On the other hand, it enables fine breaking to be carried out effectively, and thus increases the working capacity of the mill; by promoting more regular working of the latter, it also tends to yield better results generally. Moreover, except where a hillside with a suitable slope is available, it greatly simplifies and cheapens the construction of the mill proper. In flat country, and especially in establishments driven by electric transmission from some distant source of (water) power, the use of the separate rock-breaker station is more especially indicated.

In no case, however, wherever the rock-breakers may be placed, should they be run by the same motor or driven off the same shaft as the rest of the mill. The work of the rock-breaker is necessarily irregular, much more power being required when a large piece of rock is between the jaws than when small stuff is going through. This evil may be to some extent lessened by having very heavy fly-wheels, as already recommended, but, in any case, the irregular action of the rock-breaker causes any

machinery driven by a common motor or main shaft to run irregularly, and thus seriously disturbs the action of the mill where uniform driving is an important desideratum. A separate motor should therefore always be provided for the rock-breakers; it may be added that as these usually run on the day shift only, the same motor may, by a proper arrangement of shafting, be used for doing other work at night, if such is necessary.

Sorting.—In all, or nearly all mines, a certain amount of barren or low grade material, sometimes poor vein-stuff and sometimes country rock, is extracted with the payable ore. It becomes a question of great economic importance how this poor material should be treated; it is impossible to pick it all out in the mine, and it may be taken for granted that the cost of breaking it out and tramming or hoisting it to the surface has necessarily to be incurred, though it is the duty of the mine manager to see that its quantity be kept down to the lowest possible limits. Evidently it must either be crushed with the pay ore or else picked out, the latter operation being generally spoken of as sorting. The conditions that favour the former mode of procedure are:—

(*a*) Cheap power.

(*b*) Relatively cheap skilled and relatively dear unskilled labour.

(*c*) Gold contents of poor rock to be eliminated at least equal to those of the tailings ultimately rejected.

(*d*) Low cost of total range of processes required for treatment of the ore.

(*e*) Mill capacity at least equal to the total output of the mine.

The above conditions exist, for example, in California, where, with cheap water power and highly paid labourers,

it is found more profitable to mill the whole of the material stoped out of the mother lode than to attempt to pick out the rich strings of pay quartz from the very poor, partially altered country rock blasted out with it. On the other hand, the opposite state of affairs prevails at the Witwatersrand, where there is no water power to be got, the only fuel available being a high-priced coal of only moderate quality. Sorting can be done by native boys at wages, inclusive of food, of about 2*s*. 6*d*. per day of 10 hours, whilst skilled mechanics or men in charge of machinery get from 15*s*. to £1 a day. The waste rock —chiefly quartzite—which is there sorted out but rarely assays much over 1 dwt. per ton, whilst the average richness of the tailings ultimately rejected is about $1\frac{1}{2}$ dwt. The cost of extracting the gold from the ore is usually from 7*s*. to 8*s*. per ton inclusive, whilst the cost of sorting is mostly from 2*s*. to 3*s*. per ton, and at the same time the producing capacities of many of the mines, when fully developed, are frequently superior to those of their milling plants.

Under these circumstances many of the leading mines, prominent among which are the Ferreira and the Crown Reef, have adopted elaborate systems of sorting, and have been able to show in their annual balance sheets handsome profits from the introduction of this practice. The amounts sorted out at the different mines vary from 17 to 50 per cent., about 33 per cent. being an average figure. The sorting is performed either on floors or on moving tables. In the case of the former the ore is dumped over a grizzly, and is, at the same time, washed by a spray of water. The floor is covered with steel plates, and upon this the ore is turned over and the waste picked out and thrown into

cars, whilst the clean ore is shovelled into a bin. There are two forms of table in use. The first consists of a round revolving table 3 to 4 feet wide, 20 to 25 feet in diameter, driven from a central shaft or by means of a pinion and a circular rack bolted to the bottom of the framing of the table; the speed is about one revolution in 1½ minutes. The other form consists of an endless travelling belt made of steel plates riveted to a couple of chains, the width of the belt being 2 to 3 feet and its length 30 to 50 feet. Both these latter methods admit of very perfect sorting, but the wear and tear is rather heavy and the upkeep correspondingly expensive. Sorting on a floor appears to be the cheapest method, but requires a good deal of space. It is difficult to give exact figures showing the average cost of sorting, as most of the South African companies that employ this process include the cost of sorting with other items in their accounts, sorting and crushing, or sorting, crushing, and tramming to mill being usually put under one head. Obviously the cost of sorting will vary with the percentage of waste sorted out, the total tonnage treated, and the fineness to which the ore is broken before final sorting.

The following table shows the recent costs of these items in a few representative mills, being based upon the unit of one (short) ton of ore milled. From the data there given it would appear that the average cost of crushing a ton of ore as raised would amount to about 3·5*d.*, and that of sorting to about 4*d.* The former figure may be looked upon as rather high, compared with large American mines, where the crushing cost appears to be about 2*d.* to 2·5*d.* per ton.

Name of Company.	Sorting Appliance.	Tonnage milled *Short tons.*	Waste sorted out.			Cost in pence per ton of ore milled, of:		
			Actual Tonnage. *Short tons.*	Percentage.		Crushing.	Sorting.	Tramming to Mill.
				On Ore milled.	On Ore hoisted.			
Henry Nourse Gold Mining Co., Ltd.	Floors	101,100	31,597	31·6	24	6·3	9·8	
New Heriot Gold Mining Co., Ltd.	Floors	109,526	36,808	33·6	25	6·2		2·7
City and Suburban Gold Mining and Estate Co., Ltd.	Table	226,863	46,571	20·5	17·08	5·8	3·2	1·9
Geldenhuis Estate and Gold Mining Co., Ltd.	Floors	196,266	abt 65,400	abt 33	abt 25	8·20		
Crown Reef Gold Mining Co., Ltd.	Table	185,179	36,364	19·4	16·41	4·11	6·73	3·74
Paarl Central Gold Mining and Exploration Co., Ltd.	Floors	80,317	9,241	11·5	10·31	11·76		9·6
Ferreira Gold Mining Co., Ltd.	Floors	125,326	61,596	49·1	32·47	4·65	10·74	6·70*
Robinson Gold Mining Co., Ltd.	Table	180,400	23,197	12·8	11·39	4·94		1·49

* Includes: Elevating ore to upper floor of mill, 1·99 pence per ton.

CHAPTER VI

GENERAL ARRANGEMENTS—THE MORTAR BOX AND ITS ACCESSORIES—SCREENS—DIES

The Californian Stamp-mill.—The action of the gravitation stamp-mill has already been partially explained. The modern mill, the Californian stamp-mill as it is appropriately called, differs not at all in ultimate principle from its predecessor of four centuries ago, but all its details have undergone extensive modification. The old Saxon mill gradually became the cumbersome Cornish mill of the present day by the replacement of wood by iron in all working parts, thus making it a stronger, more durable, more powerful, and in every way a better machine. The Californian differs from the Cornish or improved Saxon mill principally in the following points:—The cams, instead of being short iron projections from a huge barrel, are now curved arms threaded on a spindle, the cam shaft. The cams no longer act in the median line of the stamp stem, but act entirely on one side of it, this lateral action producing a turning moment of the entire stamp upon its axis, the stamp having been made circular in section, instead of rectangular, so as to give due effect to this turning moment. The result of this alteration has been to

equalise the wear of all portions of the stamp, and thus to secure uniformity of operation. A great deal has been written and repeated in many places as to the advantage of this rotative motion in producing a grinding action on the ore in the mortar. This is an entirely erroneous idea. If the mill is kept in proper running order and well lubricated, the above-mentioned rotation takes place entirely or all but entirely when the stamp is being lifted, continuing only to a very slight extent, or not at all (depending on the adjustment of the guides), during the fall of the stamp. This rotation of the stamp is the best marked characteristic of the Californian stamp, and that which clearly distinguishes it from other types. Other minor differences are that the anvil upon which the stamps beat now consists of separate blocks, one for each stamp, instead of being one single piece of metal, or simply a bed of stone as formerly, whilst the mortar has also undergone many important modifications.

It must not be forgotten that we owe the Californian gravitation gold mill to the ingenuity of American miners, who have, by continual study applied to its improvement in minute details, gradually evolved it from the crude old-fashioned Saxon mill of the middle ages, whilst we have in Cornwall, to our shame be it said, still adhered to the cumbersome original model. And strangely enough we are doing the very same thing over again with regard to the Californian mill. This was elaborated in a country clothed with splendid forests of grand pine trees, which yielded admirable timber for constructive purposes, ready to hand at a time when iron was scarce and dear, and foundries non-existent. No wonder, then, that the American mill employed this timber whenever

possible, and had wooden frames and wooden mortar blocks. It does not, however, follow that because wood was under those circumstances by far the most suitable construction material, it should still be so, under conditions entirely different. We have slowly realised this to some extent, and iron-framed mills are coming gradually into use, but at the same time English engineers have not yet freed themselves from the trammels of imitativeness, and are still building iron mills which reproduce in iron the pattern of the American wooden ones. The sooner that this conservative policy is abandoned, and that mills are designed with regard only to the mechanical principles involved in the problem, and to the best system of solving this by the application of modern methods of iron and steel manufacture, the sooner may we hope for essential improvements in mill construction.

The Californian stamp-mill, as already stated, crushes the ore by means of the action of a heavy piece, the stamp, which, lifted by appropriate mechanism, is allowed to fall under the action of gravity upon the ore contained in a mortar. It thus consists of three essential parts :—

1. The mortar.
2. The stamp.
3. The lifting mechanism.

1. Under this head is included the mortar box proper with its screens and other attachments, the mortar block which forms its foundation, and the dies, which form, so to speak, the replaceable wearing surface of the anvil, upon which the ore is pounded.

2. The stamp consists essentially of a long stem carrying at its lower extremity a head into which is fitted a removable shoe, which constitutes the wearing face of the

stamp. With this is usually included also the tappet, which is properly speaking a portion of the lifting mechanism; as it adds, however, to the effective falling weight of the stamp and is attached to it, it is better considered here.

3. The lifting mechanism consists of a horizontal shaft on which are keyed cams acting upon the above-named tappets, and also a pulley or spur wheel, which transmits power to the shaft.

The cam shaft bearings may properly be considered here, and so may, for the sake of convenience, the guides, within which the stamp stems move.

The proper shape, proportion, and material of each of the above elements constitute essential factors in the success or failure of any mill, and will therefore have to be considered in due detail. There are also various accessories, which, though perhaps not indispensable to the working of a mill, are essential to its successful running, and which will also have to be considered; such are, for instance, the arrangements for hoisting up the stamps, and for supporting them when so hoisted, the water supply, the feeding arrangements, &c.

Mortar Block.—The mortar block is usually constructed of logs of sound, solid timber, securely bolted together. One of the best materials for it is good pitch pine, unless the weight of this timber, 1·6 times as great as Danzig pine, be considered an objection. Karri, an Australian wood (a species of Eucalyptus), close-grained and heavy, has also been used with very good results. The length of the logs is from 8 to 15 feet, the usual dimensions of the block being from 20 to 30 inches wide by 48 to 60 inches long. The logs are generally about 15 by 20 inches, and are so arranged as to break joint. They are carefully

squared and should be well tarred before being put together; a transverse strapping piece about 6 inches

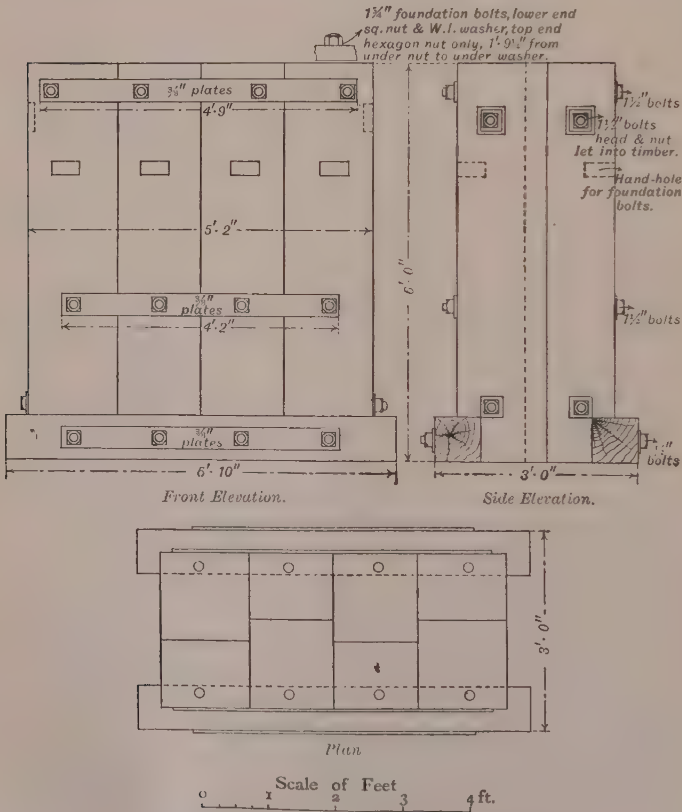


Fig. 6.

square is usually checked in for 1 or 2 inches at the bottom end of the block, so as to give a wider base and to help in holding the block together. There is usually

another strapping piece about two-thirds of the way up the block. Details of such a mortar block are shown in Fig. 6, in which the upper strapping piece is an iron plate. The mortar block should project from 2 to 4 feet above the floor level of the mill.

Mortar blocks have also been made of 2-inch or 3-inch plank, tarred and spiked together by spikes about 6 inches long. It is claimed that this makes a better block, as it is easier to get sound planks than sound logs of the above dimensions, and moreover it is considered to afford greater facilities for cutting out and replacing a defective piece. In either method the whole block is firmly held together by means of strong through bolts.

Too much care and attention cannot possibly be bestowed upon the foundations for the mortar block. It is set in a trench specially excavated for it, which should always be carried well down into solid rock, all decomposed or broken up parts of the rock being cut away and the bottom of the trench being then made as level as possible. If the sides of the trench are not wholly in hard rock, they should be protected by massive retaining walls, built of stone or brick laid in cement, or else of concrete. At least 6 inches of good concrete should next be well rammed into the bottom of the trench, being levelled off as carefully as possible; it may even with advantage receive a surface layer of strong cement in order to make all thoroughly level. Upon this foundation the mortar blocks are then set, great care being bestowed upon their alignment and their levelling. The space between them and the sides of the trench should then be rammed with concrete, or else, as is sometimes done in America, some barren quartz may be crushed in the mill and the tailings allowed to run into and fill the trench.

These tailings will pack in quite solid, aided no doubt in their settling down tight by the vibration of the mortar blocks in the partly filled trench; in either case the packing should be as thorough as possible.

The mortar blocks should have been originally cut an inch or so longer than they are intended to be finally. When they are in their place, their exact finished height is determined, and the line of their exact height carefully set out across all of them with a delicate spirit level, so that the finished blocks may all be of equal height; they are next all sawn or adzed off according to circumstances, and finally planed down accurately to this line. As a general rule the erection of the mill framing is next proceeded with, the top of the mortar blocks being meanwhile protected by a cover of planks; when all is ready for the reception of the mortar, a layer of sheet india-rubber $\frac{3}{8}$-inch thick or a couple of folds of tarred blanket are laid on top of the block. The hold-down bolts are next dropped into the holes bored for their reception, the bottom end being secured against a strong flat iron washer either by means of a cotter or a nut. Short hold-down bolts of this type are better than long bolts going far into the block, because, if one of them should break, or its thread strip, there is no difficulty in taking it out and replacing it by a new one, which could not so readily be done in the case of long bolts. The mortar boxes are then lifted into place, this being done either with strong tackle or better by means of hydraulic jacks, which are of great service in getting the box into its exact place. The nuts of the hold-down bolts are then put on and screwed down with a long spanner. After the mill has been running a couple of days these nuts should again be screwed down, putting on, if need be, two men to use the

spanner, so as to get the mortar box held down as firmly as possible. This is necessary so as to obtain a good solid bearing for the mortar, to prevent its shifting or breaking and to minimise the amount of vibration, and is a most important part of the erection of the battery, more especially so when sectional boxes have to be employed.

An improvement that will be found to be very generally applicable consists in constructing the mortar block not of wood but of a block of concrete. In all cases where the mortar blocks cannot be obtained on the spot, but have to be imported with the mill, a large saving both in first cost and in freight will result from the adoption of this method. The block must be built up from a foundation of solid rock, well levelled off, either entirely of good concrete, or else of bricks or cut stones laid in cement. Proper bolt-ways must be left for the hold-down bolts, and their heads should bear against stout plate-iron washers, which must be built into the block so as to distribute the strain as evenly as possible. Upon the block thus built is laid a 6-inch thickness of wood, best in two layers of 3-inch plank; hard wood such as teak is the most suitable for this purpose. Upon the wood again is placed, as before, a sheet of india-rubber or blanketing, and the mortar box is then bolted down as usual. The block so constructed makes a very solid and substantial foundation, and the layer of teak wood furnishes just the small amount of elasticity needed in the structure. A block for a small mill, built in this way according to the writer's design some years ago, has given every satisfaction. This plan has also been adopted in some large mills, *e.g.* at the Geldenhuis Deep, Witwatersrand, the Banner Mill, Oroville, California, and recently in the new 300-stamp mill of the Alaska Tread-

well Gold Mining Company, where a concrete block with an anvil block of cast-iron weighing about 9 tons is employed. Such a mortar block is not only cheaper than a wooden one in the first instance, but will far outlast the latter, and is far less likely to settle irregularly or otherwise work itself untrue. It is manifestly easy to build it up to the exact height required, or perhaps a very little higher, and then to adjust the exact level of the mortar box by planing down the layer of wood to the correct level.

Mortar Boxes.—Various types of mortar boxes have been used from time to time. The first Californian mills had "low" mortars, in which only the sole plate with its rim up to the bottom of the screen frame was made of cast-iron, the rest being built up of plank lined with sheet iron. The only advantages presented by this plan are the smaller first cost and lighter weight of the mortar, but these are far more than over-balanced by its disadvantages of leakiness and want of durability; moreover, as regards its effect upon the running of the mill and its crushing capacity, the lightness of the mortar is a serious disadvantage. This form may be said to have gone out of use about the year 1870, and is now never seen except under very exceptional circumstances. None of these crude forms are worth describing in detail, they having all been replaced by boxes of greatly improved construction. A fairly typical mortar box is shown in Figs. 7 to 9, the dimensions being suitable for 800 to 900 lb. stamps. It is made of good cast-iron, and cast in one piece. Much attention should be paid to the quality of the iron, which could consist (in Great Britain) of a mixture of good machine scrap, Scotch No. 3 foundry pig, and No. 3 Bessemer pig-iron. This should give a

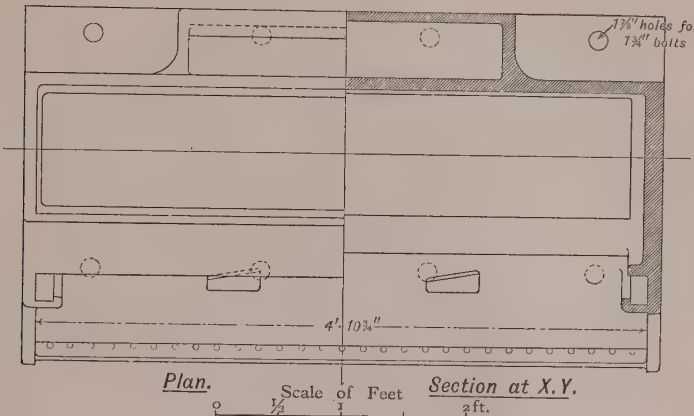
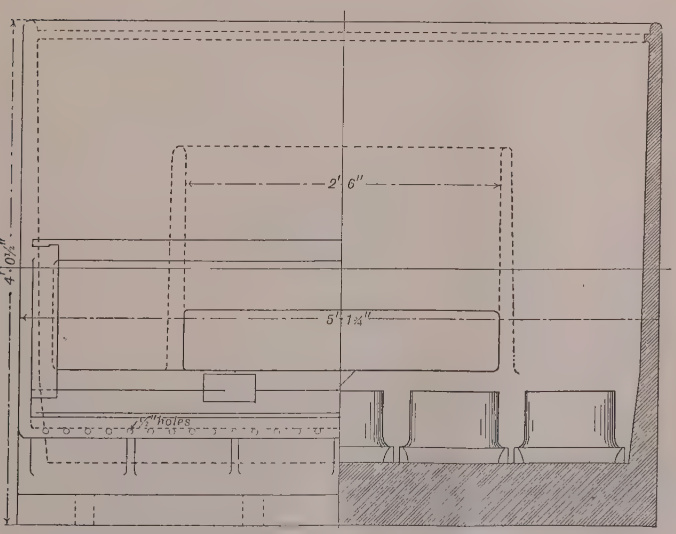


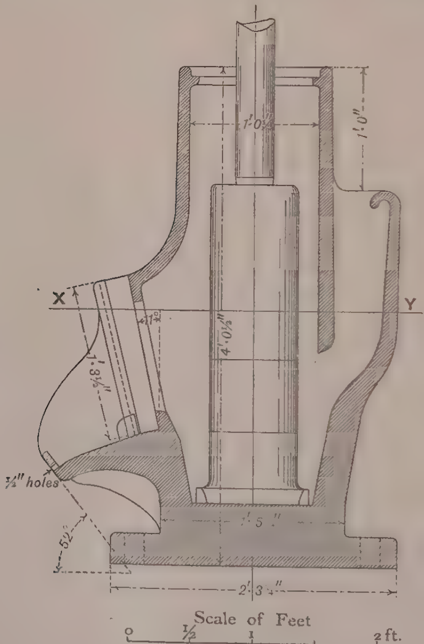
FIG. 7.



Front Elevation. right half in section.

FIG. 8.

mixture having a fairly high percentage of graphitic carbon, moderately low silicon, and phosphorus not exceeding 0·5 per cent. at most. It should be the object of the founder to produce an even and fine-grained iron, which should be perfectly soft in the thinnest parts, and



Vertical Section.
FIG. 9.

yet not show large crystalline grain in the thickest, so that a maximum of strength and toughness may be obtained, in order to withstand the vibration and strain of the continuous heavy blows to which it is subjected. After the boxes have been cast, the founder must be

careful to strip the heavier portions, such as the bottom, as soon as possible, in order that the shrinkage strains due to cooling may be evenly distributed. The box should then be tested, and if found sound sent to the machine shop. The bottom should first be accurately planed and all other work set out from the surface so obtained. The screen-frame seats must also be planed, and the holes for the hold-down bolts drilled.

The weight of the mortar box is of course proportioned to that of the stamps. The following table shows the weights usually adopted for modern stamp batteries of various sizes :—

Number of Stamps in each Mortar.	Weight of Stamp.	Weight of Mortar.	
	lbs.	tons	cwt.
5	1250	4	0
,,	1050	3	5
,,	950	2	17
,,	900	2	10
,,	850	2	5
,,	750	2	0
,,	700	1	16
,,	650	1	13
,,	550	1	10
,,	500	1	8
,,	400	1	2
3	200	0	8½

A five-stamp battery has a mortar box from 45 inches to 60 inches long, 18 inches to 24 inches wide at the base, and 39 inches to 56 inches high. In the exceptional cases where a battery consists of more or less than five stamps, these dimensions will be correspondingly modified.

It is evident that to get the best effect out of the blow of the stamp the weight of the mortar must bear a certain definite proportion to it. The considerations affecting

this ratio are so involved that it would be quite hopeless to attempt to determine it theoretically, and we have therefore to accept the above empirical data, based on experience. In order to increase the efficiency of the blow some makers are setting a cast-iron anvil block for the mortar to rest on when very heavy stamps are used.

The mortar is provided at the back with a feed shoot into which the broken quartz is delivered, either by hand or by automatic mechanical ore-feeders. The upper portion is turned over, as shown in the cross-section, in order to prevent any pieces of stone or pulp from being projected backwards out of the mortar by the action of the stamps. This feed shoot should be about $2\frac{1}{2}$ to 3 inches wide in the clear, a little less at the top, and half an inch more at the feed slot at the bottom; its position and the slope of the lower part should be such as to direct the stone on to the middle of the dies when these are in their proper position. As the lower part of the shoot is subject to much wear through the dropping on to it of the ore, it should be provided with a liner, best made of good steel plate, which can be replaced when worn out, as shown in Figs. 14 and 15. The feed shoot should be sufficiently high to prevent splashing, and may extend nearly the full width of the mortar; the slot should extend from centre to centre of the two end stamps, this arrangement favouring uniform distribution of the ore.

Screen Frames.—The front of the mortar is closed by the screens. These may be of various materials, but are always fastened to readily removable screen frames. The screen frame is best made of wood, although some few makers employ iron; this latter material is objectionable on account of its weight and because it is difficult to get as good a joint as when wood is used. The seating against which the screen frame rests should be accu-

rately planed. The frames are held in place by means of two long steel keys at either end, and one or two shorter wedges at the bottoms. Oak keys are also sometimes used. Other less usual methods of fastening the frames, such as studs or bolts and nuts, cotter bolts, or straps secured by wing nuts have been employed. Of all these latter methods the least objectionable is the use of cotter bolts, as the threads of all screws near the face of the screens are soon cut to pieces and destroyed by the scour of the issuing pulp. Yet another method is that of hingeing the screen frame at its upper edge, so that instead of being lifted out, it need only be released and swung up; this system presents few advantages and has some disadvantages, so it is not, on the whole, to be recommended. Keys as shown form by far the most satisfactory fastening. They should be made of good forged steel. They must be driven down into their places with an ordinary blacksmith's hammer of, say, $2\frac{1}{2}$ lbs. weight, and can then readily be released, provided that they are sufficiently long and have good large heads. The object is to secure the screen frame so well, that none of the pulp can find its way round the edges, but that all must pass through the screens. Screen frames should be made of strips of wood $1\frac{1}{2}$ inch thick and 3 inches broad, well mortised together; the screens should be tacked to their inner surface, and then a piece of blanket the exact width of the screen frame tacked over it. The portions of the outer surface on which the steel keys bear should be protected by having a strip of $\frac{1}{8}$ inch sheet iron screwed to them. A complete duplicate set of screen frames with screens and blanket strips ready tacked on should always be kept at hand, so that in case of a screen bursting it may be replaced with-

out loss of time. Changing screens can be done within three minutes in a mill where everything is kept in readiness. Sometimes the screen aperture of the mortar is divided into two or more divisions by vertical pieces, which form part of the casting of the mortar, with the object of stiffening the latter, small square screen frames being then used. This should be unnecessary, and is bad practice, as it diminishes the available discharge area; for the same reason, the screen frame should be made without central stiffening pieces. The length of the screen frame is, of course, determined by that of the mortar. Its height in the clear should be rather more than twice the depth of the pulp above the discharging edge of the screen frame; this depth is usually 4 to 6 inches, and an average height of 10 or 14 inches in the clear is sufficient for the screen frames of most mills. Since screens give way first at the bottom, where most of the wear is, they need only to be turned upside down when the lower part is worn out. For this reason, the top and bottom pieces of the screen frame should be exactly alike. The screen frame should not fill the whole of the front opening; there should be a space of at least 6 inches above it, so as to give access to the interior of the mortar whilst the mill is running. This space should be fitted with a piece of thin board held in place by a pair of light keys, or better still by a piece of stout canvas tacked at the top to a strip of wood, and hanging down about an inch inside the frame. This canvas can readily be lifted, and the mill man can then put his hand in to clean the inside of the screen, and take out any chips of wood, bits of rope, grass, twigs, &c., that may be floating on the surface of the pulp in the mortar, and which, if not removed, would choke the screens.

The hand holes with covers held down by bolts, that have been introduced by some makers for this purpose, are cumbersome and unnecessary.

The American practice, which has been most generally followed in England, differs from the Australian, in that the latter is to employ mostly vertical screens, whilst the former prefers them inclining outwards at a small angle, usually 9° to 15°. There are several reasons why an inclined screen is to be preferred to a vertical one. First of all, with a given height of pulp it provides a larger discharge area; then, again, it gives more room between the upper part of the screen and the stamp, and thus renders the former less liable to break, without at the same time detracting from its efficiency; the particles of quartz, too, fall more slowly down an inclined plane than down a vertical one, and, aided by the wash of the pulp, have therefore a better chance of falling through the orifices of the screen. These arguments must not, however, be pushed too far, for if the screen were to incline outwards at too great an angle, particles of crushed quartz would have time to fall back again into the mortar before they reached the screen at all. A compromise has therefore to be arrived at empirically, and experience has shown that an angle of 10° is about the most suitable for the inclination of the screen.

Splash Boards.—A splash board should be provided to keep the pulp, which issues from the screen with considerable violence, from being thrown about in all directions. Some English makers favour a sheet-iron one hinged at the top; but this is cumbersome and unnecessarily heavy, and its use is to be avoided. The mill man cannot see what is going on without lifting up this heavy cover, and he is only too likely to avoid taking this trouble. Far better and cheaper is a stout bit of canvas, hanging

vertically from the top of the screen frame. Another good plan is the use of a splash board as shown in Fig. 10, so placed that the mill man can look over it at his screens without removing it.

Screens.—Screens of various kinds have been used: they may, however, all be divided into two main classes, those consisting of plates suitably perforated, and those woven of wire.

Perforated screens of the first class may be either plain or burred; in the former case, a piece is punched clean out to make the hole, in the latter, the metal is not removed but is simply bent over inwards. The burr must always be turned towards the inside of the mortar box. The burred form has two main advantages: first, when the hole or slot gets worn larger by the cutting action of the pulp that is continually being driven through it, it can be restored to its original dimensions by laying the screen on a piece of board and beating down the burred edges with a wooden mallet, so that the life of the screen is thereby prolonged; and, secondly, the orifice widening outwards, any particle of quartz that will enter it at all can be driven through it, so that choking is to a great extent obviated. I am not aware that burred screens

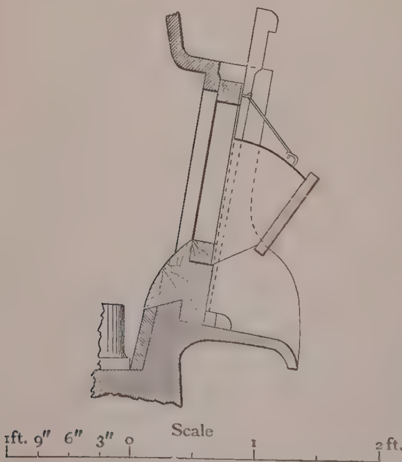


FIG. 10.

have been proved to have any corresponding disadvantage, though some suppose them to deliver rather more slowly than plain screens; it will be seen that their discharging area is as a matter of fact rather less than that of the latter type. Slot screens may have the slots arranged in various ways. Thus the slots may either break joint or have all their ends in parallel lines, the former being the better arrangement, in that it obviates a tendency of the sheet to split along the lines formed by the ends of the slots. Again, slots may be either horizontal, vertical, or diagonally arranged at various angles; all these arrangements have their advocates, the two latter being now perhaps the more common. It would seem *a priori* that the vertical slot is likely to give the readiest discharge, because particles projected towards the screens will tend to fall towards them in more or less vertical lines; theoretical reasoning is, however, of but little use on this subject. Punched holes are nearly always arranged quincuncially.

In England and Australia punched screens are usually designated by the number of holes per linear or per square inch, and slotted screens are taken as corresponding to punched screens in which the diameter of the hole is equal to the width of the slot. This is evidently an utterly barbarous system, conveying no meaning whatever, as it is obvious that the number of holes per square inch is no criterion at all, or only within the widest limits, of the actual size of the hole, which latter is the essentially important factor that determines the suitability of a screen to any given ore. The Americans designate their screens by trade numbers corresponding to the size of needle that can just pass through each; this is a better plan, but by no means satisfactory in every respect. There is in practice only one really useful way of desig-

nating screens of all kinds, and that is by the diameter of a round hole, or in case of an oval or oblong slot, of the smaller width of the aperture, expressed in decimals of an inch (or in millimetres). The length of the slot is usually between $\frac{3}{8}$ inch and $\frac{5}{8}$ inch; $\frac{1}{2}$ inch is very often adopted by American makers. Some Continental manufacturers make screens with a slot $\frac{1}{4}$ inch in length, but these are not much used. Further, the percentage of area of orifice to total screen area should always be known. The table on the following page gives generally adopted proportions for round punched, and for slot screens.

An American pattern of plain slot screens has all the slots uniformly $\frac{1}{2}$ inch long; the diagonal slots slope at an angle of 30° to the vertical and have 12 to 12·5 slots per square inch. Similar slotted screens with vertical or horizontal slots parallel ended or breaking joint have each 12 slots per square inch. Diagonal burred slot screens have only 8 slots per square inch, or about two-thirds the discharge area of plain slot screens of the same width of slot.

Of course the proportions and sizes adopted by various makers are apt to differ considerably from the above, particularly in the case of round holes.

All these screens are now mostly made of mild steel sheets, which form an excellent material for them. Russian sheet iron, which is very malleable and has a very good smooth finish, is largely used in America. The quality of the metal should be tested, as the life of a screen depends principally on the softness and malleability of the metal from which it is made. The thickness of the screen should bear a definite proportion to the width of the aperture, the relations usually adopted being shown in the following table:—

SHEET SCREENS, SLOTTED OR PUNCHED.

Needle Number.	Width or Diameter of Orifice.		Thickness of Sheet Metal.	Weight of Sheet Metal per square foot.	Slotted Screens (Fraser and Chalmers).					Punched Screens.		
					Burred Vertical Slots.		Plain Diagonal Slots.					
					No. of Slots per square inch.	Area of Discharge per square foot of Screen.	No. of Slots per square inch.	Area of Discharge per square foot of Screen.	Area of Slots taken at 0·5″ long.	Area of Hole.	No. of holes per sq. inch.	Area of Discharge per square foot of Screen.
	inch.	mm.	inch.	lbs.		square ins.		square ins.	square ins.	square ins.		square ins.
1	0·057	1·45	0·036	1·24	9·0	36·9	—	–	0·0285	0·00255	—	—
2	0·048	1·225	0·036	1·24	9·0	31·1	—	–	0·0240	0·00181	—	—
3	0·041	1·05	0·036	1·24	9·0	26·6	—	—	0·0205	0·00132	160	30·4
4	0·035	0·875	0·036	1·24	9·5	23·9	—	—	0·0175	0·00096	160	22·1
5	0·029	0·75	0·034	1·15	9·5	19·8	12·0	25·0	0·0145	0·00066	200	19·0
6	0·027	0·675	0·029	1·08	10·5	20·4	12·0	23·3	0·0135	0·00057	200	16·4
7	0·024	0·60	0·025	0·987	10·5	18·1	12·0	20·7	0·012	0·00045	240	15·6
8	0·022	0·55	0·023	0·918	11·0	17·4	12·0	19·0	0·011	0·00038	240	13·1
9	0·020	0·50	0·020	0·827	11·0	15·8	12·0	17·3	0·010	0·00031	300	13·4
10	0·018	0·45	0·018	0·735	11·0	14·3	12·5	16·2	0·090	0·00025	300	10·8
11	0·016	0·413	0·016	0·666	11·0	12·7	12·5	14·4	0·080	0·00020	360	10·4
12	0·015	0·375	0·016	0·666	11·0	11·9	12·5	13·5	0·075	0·00018	360	9·3

One of these screens will usually wear under ordinary circumstances from a fortnight to six weeks (or even more at times), depending largely on the quality of the rock being crushed, as well as on the shape and size of the mortar box. As an average for general calculations, it may be taken that one screen will outlast the crushing of 400 tons of quartz. Some mill men prefer screens made of sheet tin, the tin being first burnt off: these being thinner than ordinary sheet-iron screens do not last so long, but are supposed to discharge the pulp faster from the same size of orifice. Sheet copper is also sometimes used; this is said to wear well, but should not be used where inside amalgamation is practised, as there is risk of the screens becoming amalgamated and choking up with the amalgam collecting on them.

Aluminium bronze has been used in America, it is said with satisfactory results, its lasting powers being described as very far superior to those of iron; in this practice there appears to be no danger of the screens becoming amalgamated. The bronze employed contains 95 per cent. of copper and 5 per cent. of aluminium, and the old screens can be remelted when the holes have become too wide to admit of their further use. In spite, however, of the fact that the worn screens have thus some value, they have been found to come too dear, and seem to have gone out of use.

The cost of sheet-steel screens is between 9*d*. and 1*s*. 3*d*. per square foot, or, say, 6*s*. for a screen; the wear and tear of such screens may therefore be put down at about 0·15*d*. per ton of quartz, exclusive of carriage, &c.

Woven Wire Screens.—Wire screens differ but slightly in the shape of the mesh used; most are square, but some are slightly oblong, there being at times a difference

of from 10 to 15 per cent. in the length of the longer and shorter sides respectively. Thus a screen having 1200 meshes to the square inch is sometimes made, having 33 and 36 meshes to the linear inch in the directions of the web and the woof respectively. Wire screens have also been made having a markedly oblong mesh, the length of which is three or four times its width; it is not known whether there is any definite advantage in using this form of screen, but it would appear *a priori* that there should be, as a stouter wire could be used without diminishing the discharge area. In the case of wire screens, as in that of the punched ones, the only logical system of designation is by the width of the mesh in its narrower direction. The usual system is by the number of meshes to the linear inch; if this represents always the larger of the two numbers, whenever the mesh is not square, and if the gauge of the wire be known, the size of the mesh can be calculated. There is no absolute uniformity in the practice of the different makers, but the table on page 142 represents one of the most largely followed.

The table refers only to iron or steel wire, which latter is now very largely used. When the material is brass or copper, which are often used for the finer sizes, the wire is often of rather heavier gauge than shown in the table; for example, the following are dimensions which are usual in this country:—

Mesh Number.	Gauge of Wire (Brass).	Thickness of Wire.	Width of Mesh.
		Inch.	Inch.
30	32 S.W.G.	0·0108	0·0225
40	34	0·0092	0·0158
60	36	0·0076	0·0091
80	38	0·0060	0·0065

WOVEN WIRE SCREENS.

Mesh Number.	Number of Meshes per square inch of Screen.	Gauge of Wire.	Thickness of Wire.	Width of Mesh.		Area of each Mesh.	Discharge area per square foot of Screen.
			inch.	inch.	millimetre.	square inch.	square inch.
12	144	22 S.W.G.	0·028	0·055	1·39	0·00303	62·8
14	196	24	0·022	0·049	1·24	0·00240	67·8
16	256	25	0·020	0·042	1·06	0·00176	65·1
18	324	26	0·018	0·037	0·93	0·00137	63·9
20	400	28	0·0148	0·035	0·88	0·00123	70·6
22	484	29	0·0136	0·032	0·81	0·00102	71·4
24	576	31	0·0116	0·030	0·76	0·00090	74·6
27	729	33	0·0100	0·027	0·68	0·00073	76·5
30	900	34	0·0092	0·024	0·61	0·00058	74·7
32	1024	34	0·0092	0·022	0·55	0·00048	71·4
35	1225	35	0·0084	0·020	0·50	0·00040	70·6
40	1600	36	0·0076	0·017	0·43	0·00029	66·6
45	2025	37	0·0068	0·015	0·38	0·00023	65·6
50	2500	37	0·0068	0·013	0·33	0·00017	60·8
55	3025	37	0·0068	0·011	0·28	0·00012	52·7
60	3600	37	0·0068	0·010	0·25	0·00010	51·8

Brass and copper wires have been used where inside amalgamation is practised, and do not appear to amalgamate. Brass is not, however, a desirable metal to use for any purpose about a stamp-mill, and should be avoided. The chief objection to iron and steel is their tendency to rust; if the screens be kept in rolls, well coated with grease or composition as they leave the makers, they can be stored without fear of rusting. When a piece of the required width is cut off from the roll for use, it should be heated for a few minutes over a fire of chips till the grease is burnt off, a coat of oxide being thus formed on the wire, which acts as a good preservative against rusting.

Good wire screens will last about a fortnight if well looked after, and as they are easy to repair, sometimes even longer, corresponding to a crushing capacity of, say, 250 tons. Their cost in England is at the present time about 6*d*. per square foot. Taking the size of an average screen at about 6 square feet, this would make the cost of wire screens about 0·14*d*. per ton of ore crushed as an average figure.

A comparison of sheet and wire screens will show that the chief advantage of the former lies in their greater durability; as they are dearer than wire screens, the cost per ton of ore crushed comes to about the same thing, the wire screens needing, however, more frequent renewal, hence necessitating a little more work on the part of the mill man. The great advantage of wire over sheet screens lies in the far greater discharging area of the former, which it will be noted is for the same width of orifice rather more than 3 : 1, the difference being even greater than this in the finer sizes. This is so striking a difference that it must not be overlooked, especially in milling

low-grade ores. I have found by experiment on two batteries run side by side, one with slotted, the other with woven screens, that the latter crushed some 15 per cent. faster than the former, and this result has recently been corroborated in America. The employment of perforated screens is indicated in the case of rich ores, low in sulphurets, where inside amalgamation is practised and it is desirable not to discharge too rapidly, whilst woven screens are specially applicable to the treatment of low-grade ores, the value of which lies largely in their sulphurets (which have then to be saved by concentration) as a free discharge favours the production of granular concentrates with a minimum of slimes. Woven screens certainly seem to choke rather more readily than do burr-perforated ones, but only to a small extent, and this difficulty may easily be avoided by a careful mill man who looks after his screens well, and always keeps spare frames at hand, ready for changing.

There is no question of more importance to the successful working of a mill than that of the proper size of the orifice of the screens. Whilst something can be learnt by careful study of the ore, yet, after all, the ultimate decision as to what size of mesh should be employed is mostly arrived at empirically. There are two points to be determined in each case: first, what is the size of particle which gives the most economical result; and, secondly, what size of orifice in the screen will produce a maximum number of particles of the required size. The first point depends entirely upon the character of the ore. An ore, the main richness of which lies in its sulphurets, should be crushed comparatively coarse, so as to avoid reducing the valuable sulphurets to slimes, which are difficult both to save and to treat subsequently. If the

sulphurets are finely disseminated, the ore must be crushed more finely than when they are in coarser crystals, so as to facilitate their separation from the worthless gangue, the object being to produce a minimum of particles that can consist partly of sulphurets and partly of quartz; obviously the maximum size of the crushed particles must therefore be somewhat less than that of the valuable particles in the original ore.

Very much the same thing applies to the size of the original particles of free gold. Ore carrying coarse gold need not be stamped as fine as ore carrying free gold in a state of very fine division. The readiness with which the gold amalgamates is another factor; when the gold amalgamates very freely, it need not be retained in the battery box for as long as when the opposite is the case, whilst at the same time over-stamping must be avoided. Over-stamping, which consists in pounding the particles of gold after they have been rendered fine enough to pass through the screen, is due to a number of contributing causes, only one of which is the fineness of the screen. It is one of the most frequent causes of the loss of gold in milling. It has already been pointed out that gold when continually pounded becomes brittle, and moreover that gold in this state only amalgamates with great difficulty (page 84). Hence over-stamped gold is apt to escape past both amalgamating and concentrating machinery, being in particles so small as to float on water, and is therefore carried away and lost in the tailings.

The financial aspect of the question too must not be lost sight of; it may be that coarse crushing will cause the loss of some gold in the tailings that might be saved by stamping finer; yet if the supply of ore be plentiful and the cost of mining low, it may pay to let this gold

escape to waste for the sake of crushing a larger quantity of stone. Very often the saving of cost per ton of ore incident on crushing an increased quantity with the same staff and plant may more than compensate for the gold lost, and in such a case coarse crushing is to be recommended. The object of the mill man should be not so much to produce the cleanest possible tailings as to conduct the operations of the mill so as to secure a maximum of profit.

When chemical methods (such as cyanidation) are applied to the after treatment of the tailings with the special object of dissolving out the finest particles of gold that may be left, coarser crushing is admissible than when no such chemical process is used, because an aqueous solution can penetrate and dissolve out gold through fissures so fine that mercury, on account of its higher surface tension, could not make its way in, so as to come in contact with the gold. This is one reason why in the Transvaal the tendency of recent years has been continuously towards coarser crushing, nor is it by any means certain that the economic limit in this direction has yet been reached.

The second point also depends on a number of considerations, the size of the orifices in the screen being only one of them, the distance of the screen from the centre of the box, the width of the box, and the depth of discharge being other important factors. It is always found that, when quartz is crushed with a screen of any given size, only a very small percentage of the crushed pulp is of the maximum size, all the rest being much finer and some being reduced to slimes, which latter term is generally applied to particles less than 0·005 inch in diameter.

The following table, collected from various sources, will give a general idea of some of the results that have been obtained in investigating this subject; unfortunately the data at our command are scanty in the extreme, and more urgently needed to extend our knowledge on this very obscure point:—

Width or diameter of Orifice in Battery Screen.	Percentage of Crushed Ore.				
	Left on a Sieve having a Mesh of diameter:				Passed through a Mesh of 0·008″ diameter.
	0·024″	0·018″	0·011″	0·008″	
0·045	30	18	5	10	37
0·045	30	11	18	17	24
0·035	12	11	10	7	60
0·035	—	52	8	18	22
0·029	—	—	17	15	68
0·027	—	4	9	12	75
0·024	10	15	35	—	40
0·022	4	8	28	26	34

The percentage of slimes is thus between 25 and 75, whilst the proportion that only just passes through the screen orifices seems not to exceed 10 per cent.; it is pretty clear from this that if a screen is used rather coarser than the size to which it is desired to reduce the quartz, only about 10 per cent. will be delivered in too coarse a state, whilst about 50 per cent. will be too fine, leaving 40 per cent. in the right condition. Accordingly all that would be needed is to recrush this 10 per cent. which is too coarse. It will be seen subsequently that such coarse stuff can be separated by apparatus that is very cheap to construct and works automatically, requiring practically no attention at all; so that it is probable that in many cases the most economical way of working would be to crush through

such a size of screen as to deliver some 10 per cent. of the product too coarse, and then return this 10 per cent. to the battery, or else crush it to the required degree of fineness in some other suitable machine. Generally speaking mill men are in the habit of crushing through too fine a screen. A careful differential sifting of a parcel of tailings, each size being assayed by itself, will soon show within what limits the crushing should be conducted in each particular case. Of course it is always advisable to use as large a screen orifice as possible in order to raise the capacity of a battery to its maximum limit consistent with good work. According to Rittinger, the discharging capacity of a given screen varies as $\sqrt[5]{d^2}$, where d is the diameter of the mesh used.

In an interesting series of comparative tests made by Messrs. Morison and Bremner, the effects of varying the mesh of the screen were shown. Their results are given below in tabular form, the last column giving the tonnage that should theoretically be discharged according to the above formula; it will be seen that the figures thus calculated approximate fairly to those actually obtained.

Screen Mesh.	Measured Width of Mesh.	Tons of Quartzite crushed per 24 hours.	Calculated Tonnage on same basis.
	Inch.		
196	0·0394	9·81	9·81
324	0·0315	8·90	8·97
484	0 0240	8·34	8·05
676	0·0234	7·24	7·96
900	0·0208	7·20	7·59

Other Forms of Mortars.—The mortar previously shown in Figs. 7 to 9 may be looked upon as the

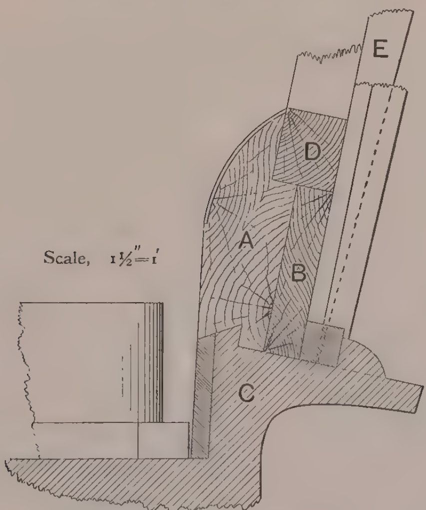
normal type of modern mortar; there are, however, various modifications of this pattern that have to be considered. Mortar boxes are occasionally made with a double discharge, back and front. In this case the front screen is arranged as usual, and a back screen about 8 inches high is fitted below the feed hopper, which is a little higher up than ordinary. This arrangement has not, however, found favour with mill men generally, and I have seen several batteries that had originally been built for double discharge, but in which the rear screen had been replaced by a piece of sheet iron, thus making the double into a single discharge box. Theoretically, double discharge should only increase the capacity of a battery whenever it cannot discharge as fast as it stamps. It would seem, at first sight, that doubling the effective discharge area ought to nearly double the crushing capacity of the box, yet the above-quoted experiments of Messrs. Morison and Bremner have shown that the screen of an ordinary single discharge mortar can pass readily 10 tons per 24 hours, so "that the screen duty of about 6·55 lbs. of ore per square inch per hour, corresponding to the above conditions, is below the limit at which the screen commences to control the crushing capacity of the mill." [1] It has repeatedly been found in practice that no better results are obtained with a double than with a single discharge mill, because in the first place the rear screen is continually giving out and requiring to be replaced by a fresh one, which can, of course, not be done without stopping the battery altogether. The rear screen is from its position more liable to damage than the front screen. With an ordinary lift of stamp (say under 10 inches) it is

[1] *Trans. Inst. Min. and Met.*, vol. viii. 1900, p. 179.

scarcely possible to so arrange the feed hopper as to deliver the rock on any portion of the die in front of the rear third; the consequence is that comparatively large fragments are thrown backwards by the blow of the stamp. Moreover, when the feed slot is raised, as it has to be in the system now under consideration, pieces of rock in dropping down will fall against the rear side of the moving heads, and are thus thrown violently against the rear screen. At the same time the rear screen is the more awkwardly situated for changing; an ore-feeder has often to be disconnected and pushed out of the way, and sometimes the men have to crawl under the feed platform to get at the screen, so that changing the rear screens takes a far longer time than changing the front ones. It is also found that on account of their inconvenient position, the rear screens are apt to be neglected by mill men and allowed to choke, so that their full benefit is not obtained even when the mill is at work. In spite, accordingly, of their apparent advantage, practical experience has led to the same conclusions as the above tests have corroborated—namely, that no benefit is derived from the use of double discharge boxes. It need hardly be said that the very fact of crushed ore remaining less time in the mortar boxes causes inside amalgamation to be less thoroughly performed. Accordingly, double discharge would not be suitable, when it is desired to catch most of the gold inside the box, and has little to recommend it under any circumstances. Screens fitted into the ends of boxes have also been tried, but have nothing in their favour, and very many disadvantages.

Inside Copper Plates.—With many ores (perhaps with most) amalgamation is best commenced inside the mortar box, and when this is done, amalgamated copper

plates are largely used inside the mortar. Usually a front plate only is employed, but sometimes there are both front and rear plates. The front plate is fastened just below the screen frame; it is sometimes attached direct to the mortar, and sometimes (preferably) to a block of wood known as the chock block, which fits just inside the screen frame. The copper plate should be 4 to

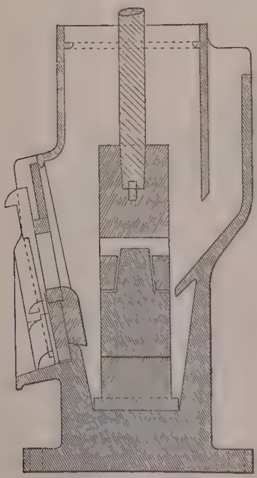


Vertical Section of Chock block.
FIG. 11.

7 inches wide, and inclined at an angle of about 40 degrees to the vertical. When it is fastened direct to the box itself the casting is arranged to receive it. Its lower end should then rest against a little step a quarter of an inch deep, and it may be secured by being slipped under little projecting lugs cast on either side of the screen seating. Another arrangement consists in drilling a number, say

about half a dozen, of holes about $\frac{3}{4}$ inch diameter, and $1\frac{1}{2}$ inches deep, into the casting; a plug of hard wood is driven tightly into each of these holes, and the copper plate is secured by means of iron-wood screws screwed into the above wooden plugs.

The object aimed at is to hold the copper plate very firmly, so that it cannot be dislodged by any accidental blow or jar, and yet to admit of its ready removal for cleaning when required.



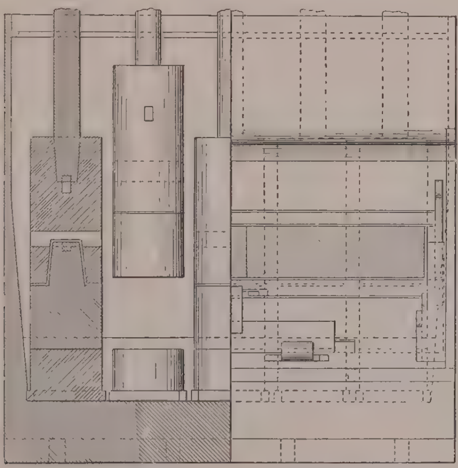
Cross Section through centre
Scale, $\frac{1}{2}'' = 1'$
FIG. 12.

The other method is shown in Fig. 11, which gives a vertical section through the entire arrangement. Here A is the chock block proper, to which the copper plate is screwed; B is a piece of $1\frac{1}{2}$-inch plank carefully planed, which makes up the full thickness of the screen frame D; E is the steel key which holds both chock block and frame in their places, and C is a portion of the solid casting of the mortar box. This is, for several reasons, probably the best method of fastening front copper plates.

When rear plates are used, a special recess must be provided for them underneath the feed shoot a prolongation of the bottom of the latter forming a sort of apron to protect the copper plate. This arrangement is shown in section in Fig. 12, which is probably the best form for

it. Fig. 13 is a front elevation of the same mortar box, which is of the well-known "Black Hills" pattern. The copper plate is held in position either by wedges or by screws, the arrangement being identical with that described for the front plate.

The use of a rear plate has certain advantages in some cases, but also many attendant disadvantages; it neces-



Scale, ½″ = 1′

FIG. 13.

sarily increases the width of the mortar box at about the level of discharge of the pulp, and this is a decided disadvantage, as it diminishes its crushing capacity. Moreover, no provision is ever made, nor would it be easy (although quite possible) to make it, by which the height of the rear amalgamated plate could be adjusted to suit the height of the dies when new and when worn respec-

tively. If the plate is so high as to be in the right place for new dies, it will be so high as to be partly above the level of the pulp in the mortar when the dies are worn down, and is then apt to become coated and useless; if, on the other hand, it is in a position to suit the worn-down dies, it will be so low as to be cut and scoured when the dies are new. The rear plate is, moreover, difficult of access, and cannot be well got at for cleaning purposes, so that, although at one time very generally adopted, its use is now largely discontinued. Both rear and front plates certainly diminish the crushing power of the mortar, but this effect seems to be less marked in the case of the front plate; at the same time, what the batttery loses in efficiency as a crushing, it gains as an amalgamating, machine. Upon the whole, the use of the front plate only is to be recommended in the majority of cases. When it is used, the mortar is some 2 to 3 inches wider at the surface of the discharge than when no plate is used. This is necessary not so much for the sake of making room for the plate itself as to diminish the violence of the scour of the pulp when set in motion by the stamps. In a narrow box this scour is so violent that all amalgam would be stripped off the plate as soon as deposited, and the plate itself rapidly cut to pieces. The width of a mortar for 800 to 900 lb. stamps should not be less than 15 inches at the discharge level, when front inside plates are used; the use of a rear plate would increase this figure by about another 3 inches. A modern mortar for 1050 lb. stamps, fitted with chock blocks, is shown in Fig. 14, this being the most recent pattern adopted by the Sandycroft Foundry Company, Limited.

In some recent batteries, *e.g.*, those of the Alaska-

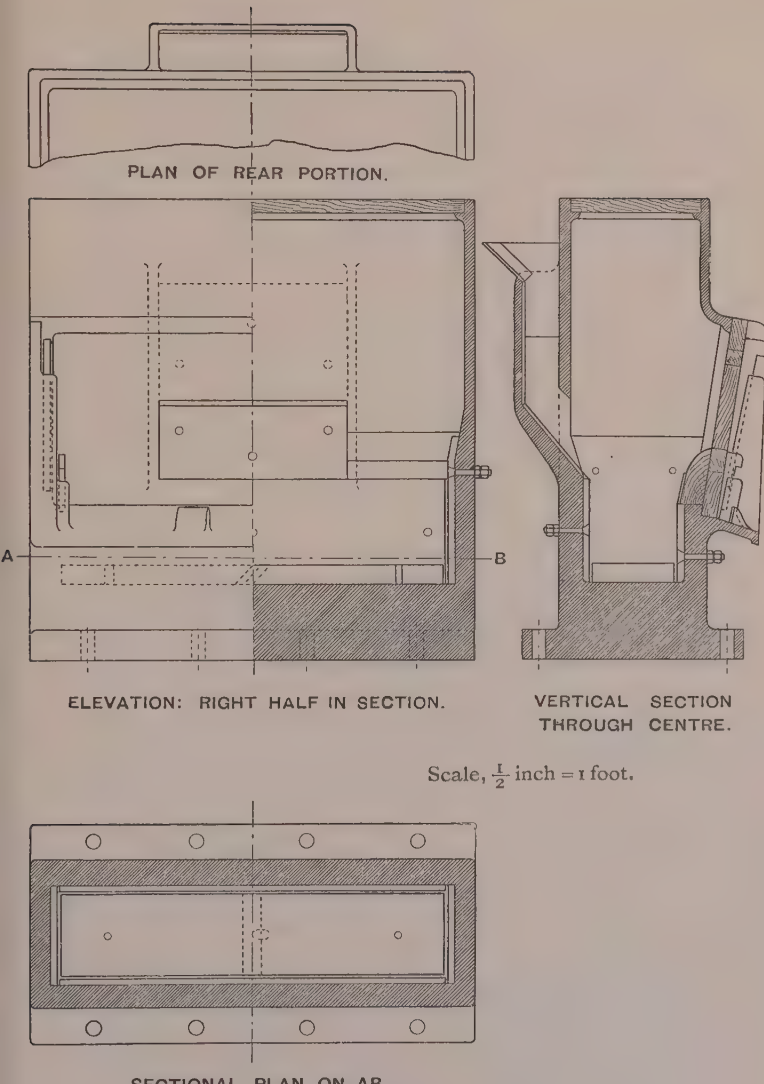
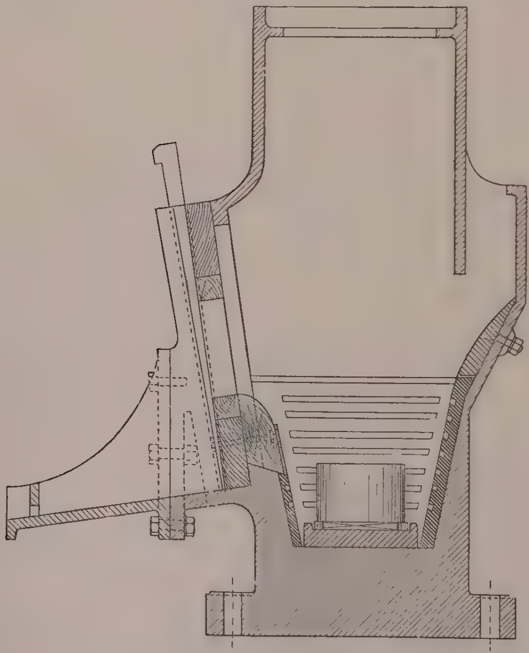

PLAN OF REAR PORTION.
A
B
ELEVATION: RIGHT HALF IN SECTION.
VERTICAL SECTION THROUGH CENTRE.
Scale, $\frac{1}{2}$ inch = 1 foot.
SECTIONAL PLAN ON AB.

Fig. 14.

Treadwell Company, Limited, Alaska, and the New Crœsus and the Simmer and Jack Mines, Transvaal, the back copper plate is replaced by riffles in the back and sides of the boxes and also at times in the front. These



Scale, $\frac{3}{4}$ inch = 1 foot.

FIG. 15.

riffles are generally grooves cast into the liner plates of the mortar (see page 159), which are then usually made of cast steel; these work quite satisfactorily, and do not need as much width as do copper plates, whilst they afford similar opportunities to the latter for the accumu-

lation of amalgam in places from which there is little risk of its being dislodged. The construction of such a mortar is shown in Fig. 15, this being the design of the Humboldt Engineering Works, Germany.

Depth of Discharge.—One of the most important factors in regulating the running of a mill is the depth of discharge, that is to say, the vertical distance from the top of the die to the upper edge of the lower portion of the screen frame over which the pulp is discharged. This varies greatly in different mills, being from 5 to 15 inches. It is obvious that a shallow discharge favours rapid and coarse crushing, whilst the opposite arrangement keeps the pulp longer in the mortar and aids fine crushing. The depth of discharge is therefore determined by the same considerations as to the nature of the ore as govern the coarseness or fineness of the orifices in the screen employed; it would be decidedly bad practice to use a shallow discharge with a fine screen, or a deep discharge with a coarse one; hence these two factors in gold milling must always be regulated so as to correspond to each other and to the character of the ore to be crushed. As the dies wear down, the depth of discharge tends to increase. In order to keep it uniform it is advisable to have chock blocks of different depths, which can be changed *pari passu* with the wearing down of the die. The total amount of the latter is usually about 5 inches; hence the mortar should be so constructed as to admit of chock blocks varying in height by this amount being inserted. If the edge of the casting which forms the lower part of the screen seating be about 1 inch lower than the top of the new die, when in its proper position, a chock block 9 inches deep will give a depth of discharge of 8 inches, which is a very usual amount; when the die

is worn down completely, a chock block 4 inches deep should be used, and the depth of discharge thus kept uniform ; if chock blocks are provided differing in depth by successive amounts of 1 inch, a quite different degree of uniformity will be attained. When in addition to chock blocks of varying depths, false bottoms are provided for insertion below the dies (see page 167), the depth of discharge can be completely and accurately controlled, and this method is now generally adopted in the most modern mills.

Inside Linings.—The only portion of the mortar box that is liable to wear is that part of the feed shoot upon which the ore drops, and those portions of the sides of the box against which the pulp is dashed by the action of the stamps. These parts should accordingly be protected by a set of steel plates from half an inch to one inch thick, cut so as to fit exactly into their places ; their vertical edges should be bevelled at an angle of 45 degrees, so that when once dropped into position, they mutually hold each other ; the sole plates of the dies, too, usually fit against them and help to keep them from shifting. It very rarely happens that one of these plates is displaced, and when this does occur, it will be found to be due either to carelessness in putting them in or else to their vertical edges being worn. The front plate should come up level with the edge of the screen seating ; if a chock block is used, this may be protected by having a piece of $\frac{3}{8}$ inch sheet steel screwed to its inner face, just below the copper plate. The back plate should reach to the bottom of the feed slot, and there should be another plate bent to the curve of the feed shoot, projecting some 12 inches or so up it, and fastened in its place by means of bolts ; the side plates should be about 9 inches

higher than the level of discharge when the dies are new. The use of liner plates allows the weight of the mortar to be considerably reduced without correspondingly shortening its life. Cast steel liner plates up to $1\frac{1}{2}$ inches in thickness are sometimes used; such plates, with grooves cast in them so as to form riffles, have already been referred to. The methods in vogue of inserting and securing liner plates are shown in Figs. 14, 15, and 16.

Covers.—The top of the mortar box should always be covered. Boxes are mostly cast with a small flange or recess about 1 inch below the top of the mortar, upon which the covers are supported. These covers are in two halves, being divided lengthways, along the centre line of the stamps. Into each half are cut semicircular recesses to allow of the passage of the stamp stems. They are best made of light $\frac{3}{4}$-inch plank, their junction being either stepped or bevelled, so as to prevent any splashes of pulp from spirting up between them. The covers may also be made of light sheet iron; in this case they should overlap about an inch along the middle of the box. The two covers may be held together by light bolts or by hooks, and should be provided with light handles to enable them to be lifted off readily.

Arrangements for Cleaning Up.—In order to facilitate the removal of the dies from the mortar box when the latter has to be cleaned up, several arrangements have been employed. One consists of a hole about $\frac{3}{4}$ inch in diameter drilled in the bottom of the box, sloping upwards from the back, and coming out in the centre of the middle die; when the battery is running, this hole is closed by a carefully turned iron pin, which is driven tightly into the hole. On cleaning up, this pin is withdrawn, when a bar can be put in and used as a drift for

forcing up the middle die. The disadvantage of this arrangement is that the hole soon gets out of true, so that the pins no longer fit it exactly, and leakage takes place in consequence. A better plan is to have a section of the front of the box made movable, opposite the middle die; this movable piece is dovetailed and carefully fitted into its place; it is held down by the screen frame when this latter is in position. When the battery is to be cleaned up, this movable piece is lifted out, and the middle die can then be prized up by means of a bar or of wedges. Care must be taken that the apron of the mortar box shall be placed so low down that any accidental leakage that may take place around this movable block shall find its way over the apron, so as to prevent loss. This arrangement is shown in Fig. 16, which represents a recent pattern of battery box made by Messrs. Bowes Scott and Western, Limited.

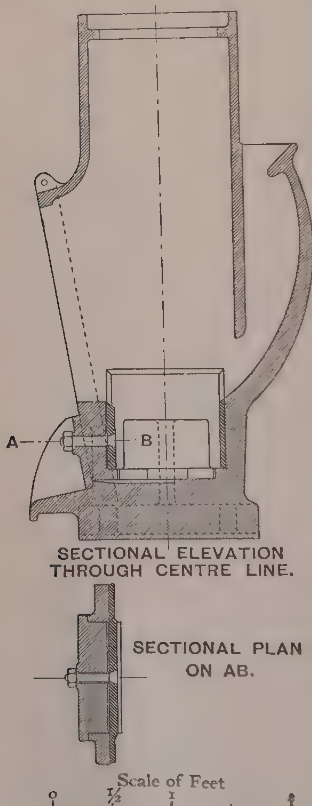


FIG. 16.

In Morison's new mortar box the entire front is left

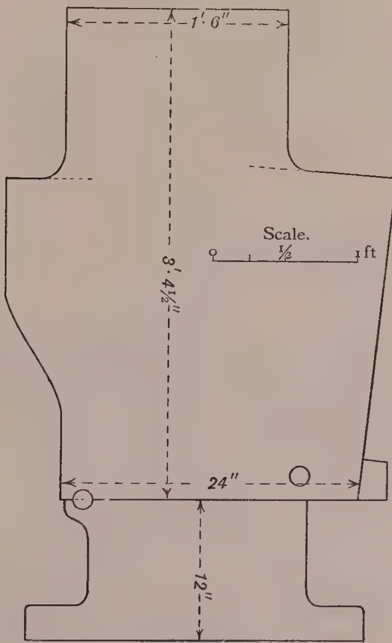
open, the front piece above the screen consisting of a removable plate pressed out of stout sheet steel, so as to give ample stiffness to the entire box, yet capable of being readily swung aside to give access to the interior.

Apron.—The apron of the mortar box is usually cast in one piece with it; at times it forms a separate casting which is bolted to the box, but this latter arrangement, which is indicated in Fig. 15, can only be recommended under circumstances where it is necessary to reduce the weight of the casting to the utmost. However carefully the joints are faced and jointed, there is always some risk of leakage, a defect which should be scrupulously avoided in a battery. The apron is exposed to a good deal of wear owing to the scour of the pulp as it issues from the screen; when therefore it forms a portion of the mortar, it should be of sufficient thickness to last as long as any other part of the box, or else it may be protected by a light replaceable liner plate. When the apron is merely bolted on, it can of course be readily renewed when worn out. The apron forms the distributing arrangement from which the pulp flows into whatever apparatus is to be next employed for gold extraction. If the pulp is to be caught in a trough or launder, the apron need have only one or two apertures through which the pulp is discharged. In the majority of cases, however, the pulp flows directly over amalgamating tables or wells, where its uniform distribution is of great importance. In these cases the front of the apron should be pierced by a number of holes about $\frac{3}{4}$ inch in diameter through which the pulp can flow, thus ensuring even distribution so long as the apron is level; to secure the greatest possible uniformity, the centres of the holes in question should be set off by the maker parallel to the planed bottom of the mortar box. This

arrangement is shown in Figs. 7 to 9. The apron should always be sufficiently wide to admit of any necessary work, such as getting out dies and shoes, being done on it, without risk of damaging the tables that may be set below it ; it ought to project at least 8 inches beyond the front of the screen frame.

Sectional Mortar Boxes.—It is sometimes necessary to construct mortars so that they may be carried into regions difficult of access, where the backs of animals, or even of men, constitute the sole available means of transport. When this is the case, the mortar has to be made in sections, so that the weight of no one piece shall exceed perhaps 2 cwt. This is under all circumstances an unsatisfactory arrangement ; however well the fitting and jointing be done, there is nearly always some leakage. Manufacturers have tried many different devices, the best so far being the arrangement shown in Figs. 17 to 19. The bottom part only of the mortar is made of cast-iron, the upper portion being of steel plate riveted or bolted together. The screen seating is also made of cast-iron. The cast-iron bottom is in vertical sections (one of which is shown in Fig. 19), held together by four bolts as shown, and by a steel dovetailed key that runs along the entire length of the box. The key-way must be planed out and the key made a thorough driving fit. The bolt holes must be carefully bored out and the bolts turned to an accurate driving fit. All the bolts should be furnished with lock nuts, so as to withstand the jar of the stamps. The various sections should be planed and faced with the utmost care, and be fitted together with breaking joint as shown without any jointing material between the sections ; if, on account of bad workmanship, it is absolutely necessary to use the latter, this is best done

by painting the planed faces with a solution of india-rubber so as to form a thin, closely adherent film of this substance on the surface. The most difficult joint to keep tight is always that between the cast-iron bottom and



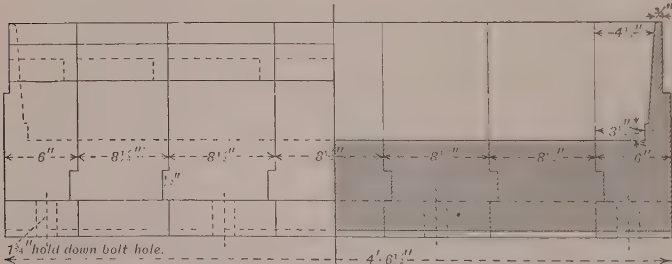
Side Elevation.

FIG. 17.

the wrought-iron or steel walls. It would probably be better (although more expensive) to make the bottom sections of cast-steel; the sides could then be riveted firmly to them and caulked, which cannot be done in the case of cast-iron. Sectional boxes are, however, always

troublesome, and should only be used in case of absolute necessity. They have, however, been used in some places with success. Thus, for instance, a forty-stamp mill with sectional mortars, built by Fraser and Chalmers, has been running for eight years at Rosario, Honduras, and has given entire satisfaction.

Dies.—The die forms, as already said, the working face of the anvil upon which the ore is crushed. Numerous different forms have been tried at various times. The die was formerly made in one piece occupying the entire



Front View of Battery Box. Right half in section, none of long through-bolts shewn.

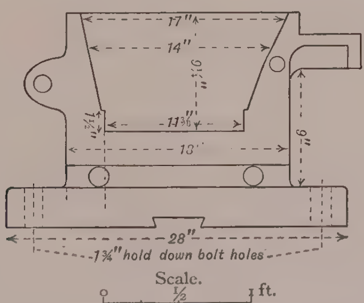
Scale.

0 ½ 1 ft.

FIG. 18.

bottom of the mortar box, but it is now always made in separate pieces, one die to each head. These have been made cylindrical, hexagonal, or octagonal, as shown in Fig. 20, a form which on account of its simplicity and suitability for forging is still at times used. Dies have also been made with lugs fitting into recesses in the mortar, but now the form shown in Fig. 21, or one closely approximating to it, is almost universally adopted. The upper portion is cylindrical, corresponding in diameter with that of the stamp shoe,

and the foot plate is hexagonal, octagonal, or more frequently square or rectangular. In this latter case the angles should be bevelled off and somewhat overhung, so as to facilitate the lifting out of the die from the mortar box. The foot plates should almost completely fill the bottom of the mortar; they should fit against the back and front liner plates, and be so set as to give the latter no play. The bases of adjoining dies may with advantage be from $\frac{1}{8}$ inch to $\frac{1}{4}$ inch apart. In



Side View of one of middle sections.

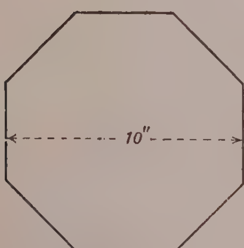
FIG. 19.

some recent mills they have been set as much as 1 inch apart to facilitate removal; this is, however, a practice by no means to be recommended, it being preferable to use one or other of the devices already described (see page 159). In any case $\frac{1}{2}$ inch should be an ample distance.

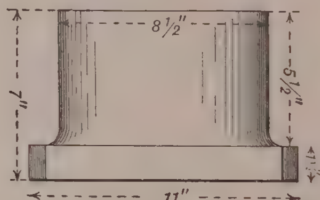
The weight of the die should be proportioned to that of the stamp, being best about 14 per cent. of the weight of the latter; it usually varies between 12 and 15 per cent. of the weight of the stamp. Thus, for a 900 lb.

stamp, 9 inches in diameter, the cylindrical part of the die should be 9 inches in diameter and 5 inches deep, weighing 90 lbs., whilst the foot plate should be 10 inches square and $1\frac{1}{2}$ inches thick, weighing 40 lbs., making the total weight of the die 130 lbs. For a modern 1150 lb. stamp-mill, the cylindrical portion of

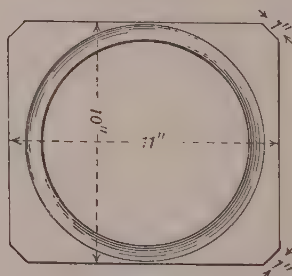
Side View.



Plan.
Scale, 1½"=1'
FIG. 20.



Side View.
Scale, 1½"=1'



Plan.
Scale, 1½"=1'
FIG. 21.

the die was made 9 inches in diameter and 6 inches high, and the foot plate $9\frac{7}{8}$ inches square and 1 inch thick; the total weight was 134 lbs., or 11·6 per cent. of the stamp. In this case a false bottom was, however, used, which allows the weight of the die to be somewhat reduced. If the die be of good material, it can be worn

down to the foot plate, so that with the above proportions, out of the total weight of 130 lbs., 90 lbs. can be used up in the mill in the former case, and 106 lbs. out of 134 lbs. in the latter. The question of the material of which the die should consist will be investigated when treating of stamp shoes, as it is evident that these two must stand in definite relations to each other. Dies must be set with all their upper faces at the same level, and are often laid upon a bed of tailings well rammed in the bottom of the mortar, about $\frac{3}{4}$ inch in thickness, and when the dies are a good deal worn, this bed may with advantage be made thicker, say up to 2 inches, according to the requirements of the mill. It is, however, a better plan to insert a cast-iron "false bottom" in the mortar to compensate for the wear of the dies; in the best modern mills false bottoms of different thicknesses are provided. These are usually made in two halves with a stepped or sloping joint in the middle, and having a drift-way cored out, so as to allow the upper half to be readily forced up for cleaning up the mortar. Such a false bottom is shown in position in the mortar box, Fig. 14.

CHAPTER VII

THE STAMP—TAPPET—STEM—HEAD—SHOE—CAM SHAFT—CAMS—CAM CURVE—POWER REQUIRED

The falling weight of the stamp determines primarily the working capacity of a mill, and batteries are therefore always gauged by this factor.

The total falling weight is made up of—

1. The tappet ;
2. The stem ;
3. The head ;
4. The shoe.

The proportion which these various portions bear to the whole weight should be about—

Tappet	14 per cent.
Stem	43 ,, ,,
Head	28 ,, ,,
Shoe	15 ,, ,,
	100

In actual practice the weight of the tappet varies between 11·5 and 16·5 per cent., the stem between 35·5 and 44, the head between 24 and 33, and the shoe between 15 and 20. A near approximation to the proportions above recommended is, however, advisable, so

that the weights for a 900 lb. and a 1250 lb. stamp should be respectively—

Tappet about	... ...	125 lbs.	...	150 lbs.
Stem ,,	... ...	390 ,,	...	550 ,,
Head ,,	... .	255 ,,	...	350 ,,
Shoe ,,	... ...	130 ,,	...	200 ,,

The practice of late years has been to increase the weight of the stem in proportion to the other parts, and in some cases to lighten the shoe. Thus in 1871 Rossiter W. Raymond[1] gave the following as the proportions customary in Colorado at that time—head : stem : collar : shoe : : 5 : 3 : 1 : 2, whereas the modern practice is given above. It must be mentioned, however, that the whole practice of milling has completely changed in the last twenty-five years.

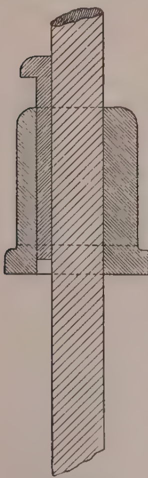
The Tappet.—Tappets have been made of very many different kinds. The simplest kind, which may for the sake of economy yet be used for light prospecting stamps not exceeding 300 lbs. in weight, consists of a collar about 2½ inches wide and some 6 inches deep, which is held in place by a vertical key, the back of the key being hollowed to fit the stamp stem.

A section and plan of this form of tappet is shown in Fig. 22, but, except under the circumstances indicated above, it is now never used.

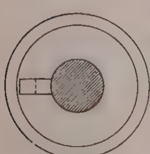
Another form, which was used in some of the earlier American mills, but abandoned about 1870, although still in use in Australia, consists of a collar into which a coarse screw thread has been cut; a similar screw is cut upon the stamp stem, the upper end of which is turned down somewhat so as to allow the tappet to slip over it. By these means the tappet can be adjusted

[1] *Statistics of Mines and Mining in the States and Territories west of the Rocky Mountains*, 1871, p. 339.

to its exact position on the stem, and can then be held there either by means of a key or of a jam nut locking down upon the tappet. This latter arrangement is shown in Figs. 23 and 24, Fig. 23 giving a plan and section of the tappet, and a plan and side view of the lock nut, and Fig. 24, a side view of the stem. It admits of very exact adjustment of the tappet, but in practice it is found that the jam nut, even when it consists, as shown, of a steel wing nut, which can be driven down by a heavy hammer, nevertheless works loose with the jar of the stamps. This arrangement has two great disadvantages: the upper end of the stamp stem has, as stated above, to be turned down, and is thus weakened, whilst at the same time it is incapable of being reversed when the lower end of the stem becomes broken or injured. The tappet also is not reversible, and has only one wearing face. An attempt has been made to remedy this latter defect by making the wearing face a removable ring of steel, which can be bolted to the body of the tappet by means of countersunk bolts. Another form of this, known as Cliff's patent tappet, made by Thompson and Co., of Castlemaine, Victoria, consists of a tapered steel body inside of which the screw thread is cut; over the outside slips a sleeve terminating in a broad flange, which forms the actual tappet face. The sleeve is held in its place by means of four set screws, the taper being, of course, downwards. None of these arrangements, which



Vertical Section.



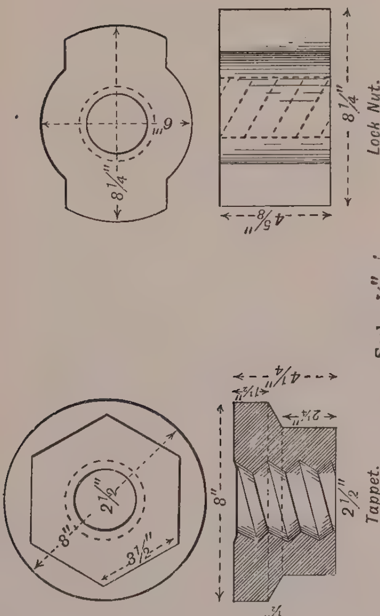
Plan

Scale, 1½″=1′

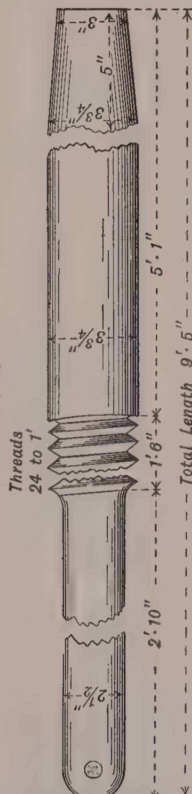
FIG. 22.

all originated in Australia, has come into general favour.

Again another system consists in making the tappet in



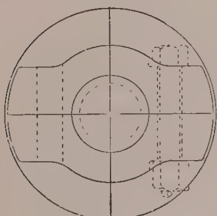
Scale 1½" = 1'
FIG. 23.



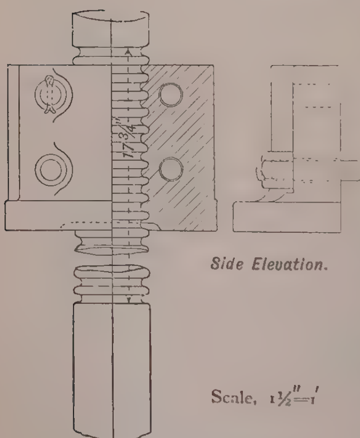
Scale, 1½" = 1'
FIG. 24.

two halves, divided vertically, these halves being held together by strong bolts, which jam it upon the stem. This tappet is shown in Fig. 25, which also shows a stamp stem with grooves turned upon it to prevent the

split tappet from slipping. If the tappet needs shifting, it has to be shifted the width of a groove, up or down as the case may be. This pattern has no special advantage to recommend it, and is, for many reasons, objectionable.


Plan.

Certainly the best pattern of tappet is the Californian gib tappet, which is now almost universally adopted.



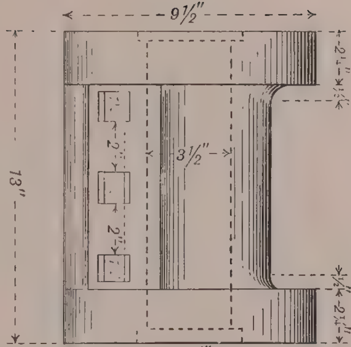
Side Elevation.

Scale, 1½"=1'

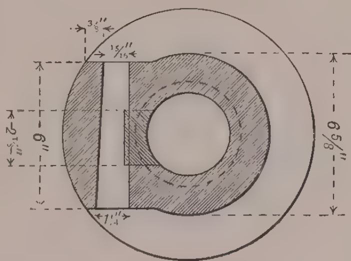
Elevation partly Sectional.
FIG. 25.

It is shown in Fig. 26 and consists of a cylindrical body having a wearing face at either end. Within a recess in the tappet is a gib made of forged steel, and this is tightened up to the stem by means of two or three tapered keys. Two keys can be used for stamps up to 750 lbs. in weight, but above that three-keyed tappets are preferable. The working face of the tappet should be equal in width to that of the cam, or even be a trifle wider, say from $2\frac{1}{2}$ to $3\frac{1}{2}$ inches. Close round the stamp stem a recess $\frac{1}{8}$ to $\frac{1}{4}$ inch wide and $\frac{1}{2}$ inch deep is turned out, the object of this being to keep the wear on the whole tappet face uniform, and to prevent its assuming any approximation to a conical

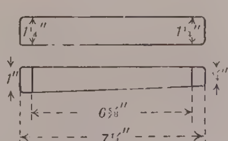
shape. It is obvious that if the working face of the tappet were allowed to become at all conical, a very severe lateral thrust would be set up between the tappet and the cam, tending to force the stamp and the tappet apart, which tendency is reduced to a minimum as long as the working faces are maintained accurately horizontal. The tappet should be accurately bored out to fit the stem so as to give a good sliding fit, the bore being less than $\frac{1}{64}$ inch larger than the stem. The gib should be of forged steel, planed on the back and edges, and the front bored to a curve about $\frac{1}{8}$ inch less in diameter than the stamp stem; this arrangement ensures a tight grip of the gib on the stem. The keys



Side View.
Scale, 1½" = 1'



Plan.
Section through centre
Scale, 1½" = 1'



Key for tappet.
Scale, 1½" = 1'

FIG. 26.

should have a gentle taper, which must be on the side away from the gib; in a vertical direction the keys should be quite $\frac{1}{8}$ inch narrower than the slots in which they work, so as to bear only against the back of the gib and the tappet, and nowhere else; their tendency to jam or to burst the tappet is thus reduced to a minimum. These keys should be left rough top and bottom, and only planed and fitted where they bear against the gib and the corresponding back portion of the tappet. When the mill is set up, these keys should be driven well home with a $2\frac{1}{2}$ lb. blacksmith's hammer; they will then hold the tappets firmly in place without any slip, and there will be but little danger of bursting the former. Some makers supply the tappet with a set screw to hold the gib firmly in place whilst the keys are being driven home; the screw is removed before the stamp is set to work. Tappets are usually made of cast-iron; this should be good, tough, close-grained iron of the best quality. Chilling the wearing face of the tappets has been tried but cannot be recommended, as the edges are apt to chip. Many modern mills have been made with cast-steel tappets; this, of course, is a better although a more expensive material, and while its adoption may be advised for very remote localities, freights to which are high—especially in the case of proved valuable mines, where the first cost of the mill becomes a comparatively secondary consideration—it is by no means a necessity, as good cast-iron tappets will wear for years without renewal.

The Stamp Stem.—The stamp stem forms the main portion of the entire stamp. The recent tendency has been to increase its diameter so as to make it as stiff as possible. A long thin stem is apt to spring in working, causing excessive friction in the guides and rapidly wearing them

away, besides increasing the liability of the stems to break; a stout stem is free from these objections, and in addition gives a more truly vertical and more effective blow. The diameter should therefore always be as great as possible within reasonable limits; this will necessitate the stems being kept as short as possible, but it is evident that there is a minimum length below which the stem cannot be reduced. In practice the stamp stem varies from 9 feet to 16 feet long, and is from $2\frac{1}{4}$ to $3\frac{3}{4}$ inches in diameter. Good dimensions for a 900 lb. stamp stem are:—

Length	11 feet.
Diameter (rough)	$3\frac{3}{4}$ inches.

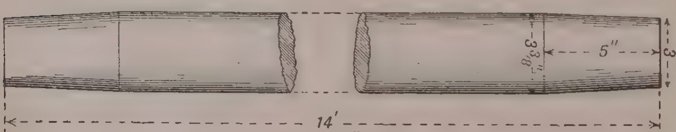
The material should be either wrought-iron of first-class quality, preferably good hammered scrap, or else Siemens-Martin or Bessemer steel, very low in carbon, say about 0·2 per cent. The stems may be either cold rolled to gauge or else turned all over. The former is slightly the cheaper method, but not by much, because the stems have to be put into the lathe to have their ends tapered down, and once centred, it does not take long to take a finishing cut off the entire length of the stem.

When the screw tappet is used, the stem must, as already explained, be turned down at one end, and have a screw thread cut on it, as shown in Fig. 24; the other end of the stem is then tapered for driving into the head. When the ordinary pattern of tappet is used, the stem is of the same diameter throughout, both ends being tapered down, as shown in Fig. 27, to fit into the head. This taper should be slight, preferably about 0·5 inch to the foot.

When a large mill is being constructed it is advisable

to turn up a standard stamp end to the exact size required; a female gauge to fit this exactly is then bored, and these two standard gauges are kept in the machine shop, all the stems being turned and all the heads bored to them, as well those required for the original mill as the spare ones required from time to time to replace worn out or damaged parts. By this system the great advantage will be secured that all stems and heads will be strictly interchangeable, and that new parts ordered, even after a lapse of several years, will be sure to fit exactly.

Stems are not as a rule worn out. They are apt to break after some years' wear, owing, according to some



Scale, 1½" = 1'

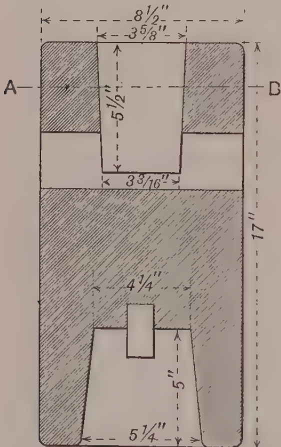
FIG. 27.

authorities, to crystallisation of the iron, induced by the continual jar; it has, however, never been proved that iron can thus be crystallised by concussion, and it is difficult to see why, if this explanation be correct, they should so regularly break in the same place. In any case a high-class soft steel seems to give even better service than the best wrought-iron. When stems break, they do so, as a rule, close to the head; the broken stem can then be inverted and used again. When both ends are broken, fresh pieces can be welded on, or the stems turned down again to fit the heads, so that stems may almost be said to wear an indefinitely long time.

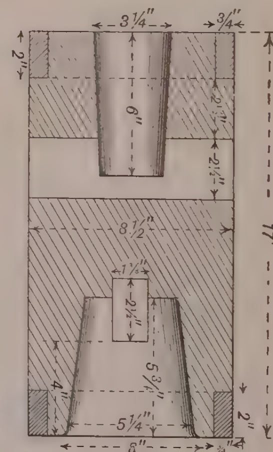
The Head.—The usual, it may almost be said the uni-

versal, pattern of head, or boss as it is sometimes called, is shown in Fig. 28. Its length varies as a rule between 16 and 20 inches, its diameter being the same as that of the shoe proper. It consists of a cylinder of iron, one end of which is bored out to receive the tapered end of the stem, and the other recessed to receive the shank of the shoe; this latter socket need not be bored out; it can be cast with quite sufficient accuracy for the purpose, and is, in fact, better left rough, as it thus holds the shoe more firmly. There are a couple of drift-ways through the head to admit of the insertion of a tapered steel drift, by means of which the end of a stamp stem or the shank of a shoe can respectively be driven out. These drift-ways are sometimes parallel, but more often at right angles to each other, the latter arrangement weakening the head less than the former one. Some makers shrink a hoop of wrought iron round the top or the bottom end of the boss, or sometimes round both ends, as shown in Fig. 29. It is quite unnecessary to hoop the upper or stem end of the head; the shank of the shoe is so much larger than the end of the stem, that there is necessarily much less thickness of metal round the former than round the latter. Hence, if the lower end is sufficiently strong to stand the driving home of the shoe shank even when strengthened by a hoop, the upper end can certainly dispense with such strengthening, and the upper edge of the casting is preferably neatly rounded. It is by no means necessary to have a hoop even round the lower end of the head, but if it is thought desirable to use one, care should be taken that the casting projects $\frac{1}{8}$ to $\frac{1}{4}$ inch below the hoop. By this means, if the top end of the shoe batters against the bottom face of the head, as it will sometimes do, the

edge of the latter will burr up a little and hold the hoop in its place, whereas if the hoop were to project the least bit below the casting, this same battering would

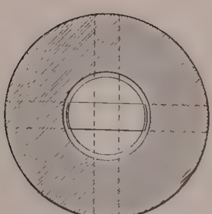


Section.



Section.

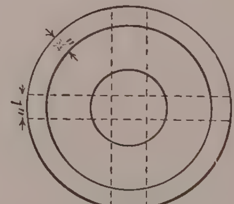
Scale, 1½″=1′



Section at A B.

Scale, 1½″=1′

FIG. 28.



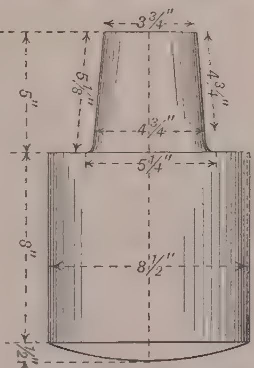
Plan.

Scale, 1½″=1′

FIG. 29.

enlarge its diameter, and it would work loose. It must not be forgotten that a loose hoop is much worse than none at all, seeing that the casting has been turned down

to receive the hoop, and is accordingly weakened by that amount. I am of opinion that hoops are best dispensed with altogether, and that the metal of which the heads are made should be of such quality as to withstand the bursting tendency of the conical pieces which have to be driven into either end. The material of the head is usually cast-iron; it should be of first-class quality, low in phosphorus and silicon, and of close, even texture throughout. No English iron should be used in the mixture. Of late years there has been an increasing tendency to make stamp heads of cast-steel instead of cast-iron; the difference in price is not very great, and there is of course far less liability to fracture. However, even a good cast-iron head will last for many years, as there is no tendency to wear except in the sockets; heads usually continue in use until they are split, an accident which is generally caused by driving in shoe shanks a little too large for their sockets.



Side View.
Scale, 1½" = 1'
FIG. 30.

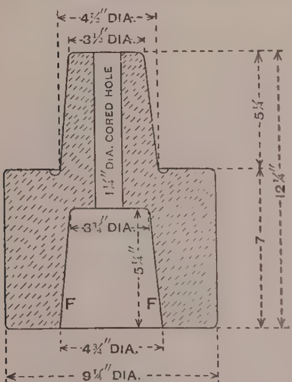
The Shoe is practically always of the shape shown in Fig. 30, from which there may be said to be no departure. It may be looked upon as consisting of two portions, the shank and the butt. The former is the part that enters the head and holds it in its place; the latter is practically the wearing face of the entire stamp. Sometimes the shank is made hexagonal or octagonal in section; this practice, formerly more common, has of late years been almost abandoned in favour of the simpler, more

convenient, and in every respect better, circular shape. The length of the shank is usually between 4 and 6 inches. Its taper is a matter of some importance; if too slight the shoe will not be held firmly in the socket, and if too great there is considerable risk of splitting the head. The taper shown in the figure is very suitable and works well in practice. It may be taken as varying from $1\frac{1}{4}$ to $1\frac{3}{4}$ inches per foot. As a general rule the diameter of the base of the shank is half that of the butt, and the diameter of the narrow end of the shank is about two-thirds to three-fourths of that of its base. The shank should not form a sharp angle with the butt, but should be neatly rounded off so as to diminish the liability to fracture at this point. The diameter of the butt is of course the same as that of the head, and the die is usually the same, though some makers seem to prefer the latter a trifle larger; this plan is not, however, to be recommended. The diameter of the butt of the shoe is an important point, as the efficiency of the stamp depends upon it; it should be strictly proportionate to the total falling weight of the stamp and the character of the ore. Thus a 900 lb. stamp for crushing hard quartz has usually shoes 9 inches in diameter, although for softer stone this may be increased to 10 inches; the effective weight of the stamp may therefore be taken as between 14 and 11 lbs. per square inch of crushing surface. This figure may be made a trifle less for lighter stamps, so that the diameter of the shoe of a 750 lb. stamp should be between $8\frac{1}{2}$ and $9\frac{1}{4}$ inches, and of a 600 lb. stamp between $7\frac{1}{2}$ and $8\frac{1}{4}$ inches. The above calculations are for low drops averaging, say, 6 inches; if a higher drop be used, increasing the momentum of the stamp, the above-given diameters may be slightly

exceeded, whilst for a 3-inch drop, the diameter, even for a 1,250 lb. stamp, should not exceed 9 inches. The depth of the butt usually varies between 5 and 7 inches. The only advantage of a deep butt is that the shoe will run longer without requiring to be renewed. If we assume, for the purpose of simplifying the conception, that the shoe can be worn right down to the shank, it is evident that, with a deep butt, the proportion of useful to useless metal (as far as wear is concerned) will be greater than with a shallow one, so that there is an apparent economy in the use of deep shoes. But, on the other hand, with a deep butt the difference between the effective weight of a stamp when the shoe is new and when it is worn out, will be proportionately greater, and the efficiency of the stamp is thus seriously impaired. This is a far more important consideration than the former one, which becomes, indeed, of very little moment in the case of big mills which have a foundry attached to them for the purpose of casting their own shoes. So much is this the case, that such mills would probably find it advisable to diminish the depth of the shoe even considerably below the minimum figure given above. On the other hand, mills that have to import their shoes from a great distance, will find a deep shoe the more economical. Thus, on the Rand, shoes with butts 9 inches in diameter and as much as 12 inches deep are in use, the shanks being $5\frac{1}{4}$ inches high, and $4\frac{1}{2}$ inches diameter at the base, tapering to $3\frac{1}{2}$ inches, and weighing altogether 240 lbs., of which quite 200 lbs. can be worn off in work.

The shoe shown in Fig. 30 weighs when new 148 lbs.; of this the butt weighs 128 lbs. and the shank 20 lbs. It is scarcely ever possible to wear the shoe down until the depth of the butt is less than $\frac{1}{2}$ inch; indeed this

result can only be obtained with exceptionally good material. As a rule, shoes are used until there is less than an inch of the butt left, when they are thrown aside and new ones put in. The alteration in the falling weight of the stamp caused by this wear of the shoe is thus quite sufficient to make a marked difference in the efficiency of the mill. In the case of a 900 lb. stamp using the shoe above illustrated, the loss of weight of the stamp owing to this cause would amount to about 110 lbs., or 12 per cent. As it is important to keep the falling weight approximately uniform, several devices have been employed for this purpose. One of the best consists in the employment of a "false shoe" or "chuck shoe," which is secured in the head just like an ordinary shoe, while the partly worn down shoe is in its turn driven into the former. A section of such a false shoe, as made by Messrs. Fraser and Chalmers, Limited, is shown in Fig. 31. False shoes of several different depths, increasing by, say, $1\frac{1}{2}$ inches at a time, should be provided. Another plan is to have at hand sets of heads of various depths, and to put on deeper heads in proportion as the shoes wear. This latter plan is open to the objection that the heads have to be so often put on and taken off, that they are apt to wear loose. A suggestion of my own, put forward some years ago, to key discs of iron,



Scale, $1\frac{1}{2}$ inches = 1 foot.
FIG. 31.

slotted so as to pass the stem, above the tappets or above the heads, where they could be held on by set screws, has never, as far as I know, been adopted in practice, though it presents the advantage that neither heads nor shoes need be removed until entirely worn out; the tappets would, however, have to be shifted to keep the drop uniform.

The shoe is fastened into the socket of the head by means of wooden wedges; these should be made of dry, seasoned, white pine, sawn into shape, but not planed. They are usually $\frac{3}{8}$ to $\frac{3}{4}$ inch in thickness, 5 to 6 inches in length, according to the depth of the shank, and tapered to suit the taper of the shank. Thus, for the shoe shown above, the wedges would be $1\frac{1}{4}$ inch wide at the base, tapering to 1 inch. It is evident that the width of the wedges, at their top and bottom, must be proportional to the diameters of the top and base of the shank respectively. These wedges are laid round the shank and tied there. A number of wedges, ready made up into "bracelets" by tying each wedge to its neighbour by means of a bit of string passed round each in succession and knotted, should always be at hand in the mill. Whenever a shoe drops out of the socket, one of these ready-made "bracelets" can be slipped over the shank, so as to secure it in its place again without loss of time. As the sockets are apt to vary a little in diameter, "bracelets" made of wedges of various thicknesses should be kept in stock.

In setting up a stamp, the head is first placed in position upon a piece of 3-inch plank laid upon the dies, which latter are supposed to be already in their places. The stems are then dropped in and tightened by a few blows from a heavy sledge hammer on their upper ends,

a piece of board being interposed to keep the top of the stem from being battered. Unless the stem-sockets are much worn, it is best to drive the stem directly into the head. When the fit is a bad one, a piece of canvas or of thin-sheet iron cut to fit the socket exactly may be wrapped round the end of the stem, but it is far better to dispense with anything of the kind. The stem is then hoisted up, and the shoe, with its wedges tied on, placed on the plank, the stem and head together being then dropped over it; by dropping the whole several times on the plank, the shoe is driven home. As soon as the wedges are thoroughly wetted by the battery water, they expand, and hold the shoe very firmly in its place.

The plank is then taken out, and a block equal to the desired length of drop is set on each die. The tappet is then slipped over the stem with its gib in place and allowed to slide down till it touches the point of the cam, the latter being in its highest position. The tappet keys are then driven well home, and the stamp is allowed to drop gently a few times till every part has been forced into its place.

Material of Shoes and Dies.—This is now either hard cast-iron or forged or cast steel. The production of the two latter kinds is a rather special trade, and it is rarely that any mill, however large, would be in a position to produce its own. Large mills in remote districts, having a foundry attached, will therefore generally find it advisable to use cast-iron. The butts of the shoe and die should be cast in heavy chills, the thickness of which should not be less than two-thirds of the diameter of the butt, whilst the shank of the shoe and foot-plate of the die should be moulded in sand, thus keeping the material of the latter portions as

soft as possible, whilst the former, if a good chilling mixture of iron be employed, will be hard and white throughout. There may be a small amount of mottling near the junction of the two grades of metal, but there should not be much, and a little will do no harm. Cast-steel shoes of open-hearth steel containing about $\frac{1}{2}$ per cent. of carbon have been largely used, but better results have been obtained of late years by the use of special metals, such as manganese- and chromium-steel. Both of the latter seem to be hard, tough materials, admirably adapted for the purposes of the stamp-mill. Forged-steel shoes have been a good deal used, but were found to wear very irregularly, and were hence some years ago almost abandoned in favour of cast-steel; but quite recently Messrs. Fraser and Chalmers, Limited, have led the way in introducing improvements in their manufacture which have again brought them to the front. At present the position seems to be that special cast-steel shoes are certainly harder than even the best forged steel, and are thus worn away more slowly than the others, but, on the other hand, are less uniform in structure and more apt to show blow-holes, which may cause them to chip. It is very likely that there will always be more or less divergence of opinion as to the really best material for shoes and dies, but it may be taken as pretty certain that the best material, irrespective of cost, is either first-class forged steel or else a high quality of special cast-steel, and that there is not much to choose between them. Some engineers assert that the best arrangement is to combine a steel shoe with a cast-iron die, and that the wear of each is thus rendered more uniform, and it would seem that there are substantial reasons for supposing this to be so. Probably a forged-steel die with a chrome-

or manganese-steel shoe would form a very satisfactory combination. The conditions of wear of the shoe and die are slightly different, for whereas the face of the die is always protected by a layer of quartz, the shoe, on the other hand, is always pounding upon this substance. It has already been pointed out, page 11, that quartz is necessarily harder than steel; hence abrasion of the latter must take place, and the consequent wear of the shoe is greater than that of the die. This used to be far more marked in the older system of working, when it was customary to keep a comparatively deep layer of quartz over the dies, than it is with the modern system of shallow stamping. Broadly it may be said that about twice as much metal is now worn off the shoe as off the die in the same time.

The life of the shoes and dies depends upon so many different circumstances that it is scarcely possible to give any accurate data; it seems that cast-iron wears one-third to one-half as long as chrome steel in most instances, but this has not been quite a universal experience. The consumption of metal per ton of quartz crushed is the best basis of comparison, since the length of time that a shoe or die will wear of course depends upon its dimensions as well as on the quality of its material and also upon the crushing capacity of the mill. Good cast-iron shoes will lose 0·4 to 1·5 lbs. of iron per ton of quartz crushed, and good steel 0·3 to 0·7 lb., but these figures are apt to vary very widely with the quality of stone to be crushed and the degree of fineness to which it has to be reduced. Adding the amount lost similarly by the dies, the average loss of cast-iron may be put down as 1·5 lbs. per ton, and of steel 0·5 lb. to 0·75 lb., as a rough estimate. In some recent comparative tests in South

Africa, it was found that good forged-steel shoes and dies lost 0·30 lb. and 0·21 lb. of metal respectively per ton of ore, the corresponding figures for cast chrome-steel being 0·29 lb. and 0·16 lb. A reduction of an inch in the length of the $8\frac{1}{2}$-inch diameter shoe shown in Fig. 30 corresponds to a loss of weight of about 15 lbs., the total amount available for wear being 110 lbs. Every mill man ought to weigh his worn-out shoes and dies and to note carefully the dates on which he has to put in new ones, and thus determine exactly the consumption of metal per ton of quartz crushed. The best metal to employ depends upon local circumstances. A large mill, casting its own shoes and dies, or a small one, in the immediate neighbourhood of foundries that will buy back the worn-down shanks and foot-plates, will probably find it more economical to use cast-iron, but a mill situated in a remote district where the high price of transport is the main factor in the cost of supplies, should use only the best steel that can be procured in spite of its higher first cost. It is worth noting that a good mine smith can make excellent quartz-breaking sledges from the shanks of worn-out forged-steel shoes.

Lifting Mechanism.—The portion of the mechanism of the battery by which the lifting is done consists of the cam shaft, upon which are threaded the cams, together with the driving pulley or spur wheel; and together with these may be considered the cam shaft bearings. It may be noted that the old European system of a cam barrel of large diameter with short projecting cams has been utterly abandoned.

The Cam Shaft is usually made sufficiently long to drive the stamps of two batteries, though sometimes a short cam shaft driving only one battery is employed. The

advantage of the latter method is that each battery can be stopped independently of the others, in case repairs of any kind are required; its disadvantage is that it entails the expense of a double number of drivers, that it takes up rather more room, and necessitates more wearing parts. It can only be recommended for a small mill of less than twenty heads of stamps. The cam shaft has to be both strong and stiff; it must be able to transmit the full power required to work the battery without any perceptible deformation due to torsion, and must be able to support the weight of all the stamps without bending or springing in the least. Theory and practice both combine to point out that a cam shaft for ten heads of 900 lb. stamps should not be less than 5 inches in diameter, and is best made 5½ inches; if working only five heads, it need not exceed 4½ inches in diameter. For heavier stamps (over 1100 lbs.), these figures are best increased to between 6 and 6½ inches for a ten-head, and 5½ to 6 inches for a five-head cam shaft. The best material for the cam shaft is undoubtedly cast steel, preferably open-hearth. Wrought-iron may also be used, and in that case it must be the best hammered scrap; steel is however almost always used for it. It should be carefully turned to gauge, and if a large mill is being built a standard pair of gauges of the exact diameter of the finished shafts should first be made and kept in the machine shop, as all the cams will have to be bored out most accurately to this size.

Drivers.—In the usual American method of mill construction, the cam shaft is driven by pulleys and belting off the main lay shaft of the mill. Most Australian engineers seem to prefer to place a heavy spur wheel on the cam shaft, and to drive by means of pinions on the

lay shaft, which must then be close to the cam shaft. Each system has its own advantages; whilst, on the one hand, it is at times awkward to get the shafts near enough together for the use of gearing, it is also at times difficult to place them far enough apart for the proper use of belting. The cost of maintenance of belting is considerably higher than that of gearing, but on the other hand, should any accident happen to stop the stamps suddenly, the belt will slip and injure nothing, whilst in the case of gearing, either the teeth of the latter or else the motor driving the mill may be seriously damaged; this last objection may be partly overcome by the use of friction gearing or friction clutches.

There is thus no decided advantage in either method; the adoption of one or the other will depend to a great extent upon the character of the mill framing and the disposition of the mill as necessitated by the physical features of the mill site. Upon the whole, however, most engineers seem to incline to the American method of pulleys, without perhaps any very sufficient reason.

Pulleys.—As there is no outside bearing to a cam shaft, the cam pulley overhangs its bearing, and it is therefore important that it be made as light as possible, consistent with proper strength. The usual size for these pulleys is from 4 ft. 6 in. to 7 feet in diameter, and the width of the pulley-face will usually be between 18 inches for a heavy ten-head battery, and 6 inches for a light five-head one.

Cast-iron pulleys are not suitable for this work; they are very heavy, and the arms, unless exceptionally strong, are apt to be cracked by the vibration of the machine. Wrought-iron pulleys with iron or steel arms are also unsuitable, because the arms are frequently

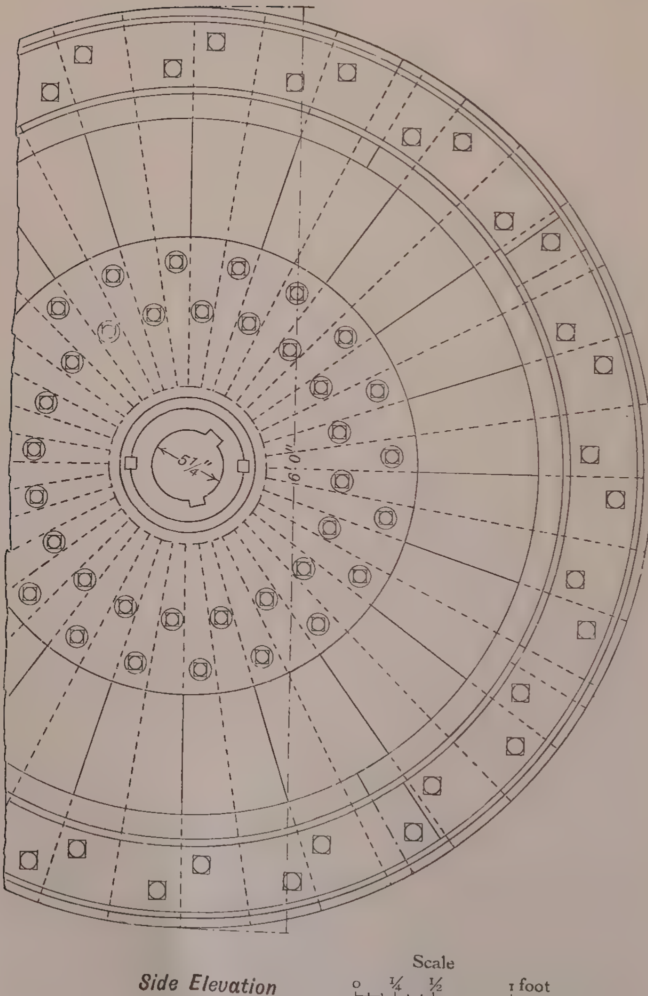


FIG. 32.

found to break for the same reason. If for any cause it be considered advisable to use a wrought-iron pulley, the arms should be made of specially low carbon steel, so as to be as soft as possible, and they should be shrouded on either side up to the rim with 1-inch planking completely filling up the pulley, so as to lessen the vibration as much as possible. Pulleys thus protected have been run successfully. The best class of pulley, however, is the American built-up wooden pulley on a cast-iron boss. This is shown in detail in Figs. 32 to 34.

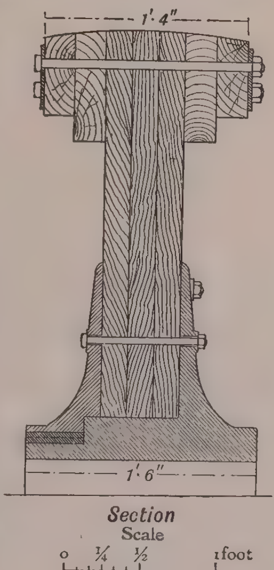


Fig. 33.

The wood should be well-seasoned red deal, or preferably pitch pine; all joints should be tarred, and the planks of which it is built carefully spiked and bolted together.

The cast-iron boss is in two pieces, one forming the sleeve which keys on to the cam shaft, the two discs, one loose and one forming an integral portion of the sleeve, being drawn together by numerous bolts so as to hold the wooden plates securely. These pulleys are usually first put together and keyed into place, and their rims are then finally turned up on the shaft itself to ensure their running perfectly true. Belts grip well on the wooden rim, and the composite wooden pulley is in every way an excellent and a durable one, if care is taken to give it an occasional coat of paint. The only objection to

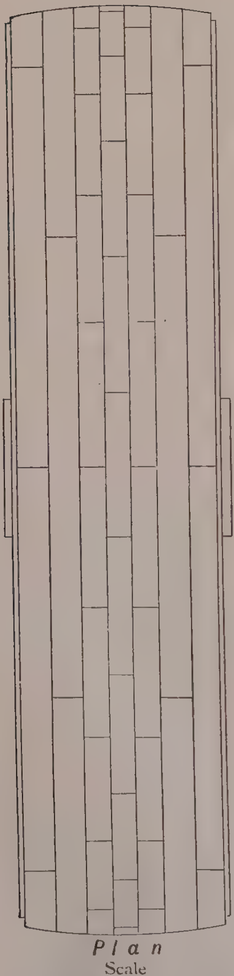
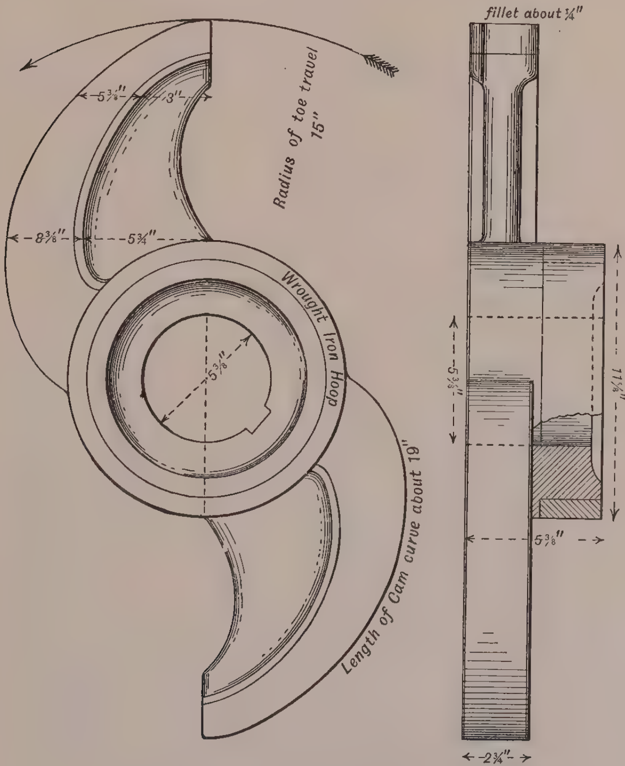


FIG. 34.

it is its great weight. The pulley illustrated in Figs. 32 to 34 weighs 18 cwt. 2 qrs., out of which the two parts of the boss weigh respectively 5 cwt. 3 qrs. and 3 cwt., the woodwork weighing 9 cwt. 3 qrs.

Gearing.—When gearing is used, it usually takes the form of a spur wheel 3 ft. 6 in. to 5 ft. 6 in. in diameter on the cam shaft, driven by a pinion of one quarter to one half its diameter on the lay shaft. It is decidedly advisable that both these wheels should be made of cast-steel, but in any case the pinion should be, even if the wheel is made of cast-iron for the sake of economy. The arms should be strong enough to withstand the jar of the stamps as well as the direct strain of the transmission of the power; the rims of both wheels should be shrouded as far as their pitch circles; the teeth should be made very strong, and should preferably be helical V teeth or else curved ones, so that their action shall be quite continuous, whilst backlash when thrown sharply into gear is avoided. The pinion is generally operated by a sliding

clutch of the usual kind, but friction clutches can be used with advantage. Too much care cannot be be-



Side View. Scale 1½"=1' *Front View.*

FIG. 35.

stowed on the material and finish of the spur wheels, and it is best that they should be machine-moulded and carefully worked up to their true shape so as to run with

a minimum of lubrication, the necessity for the employment of which is one of the objections to spur gearing.

The Cam.—Cams have been made one- two- and three-

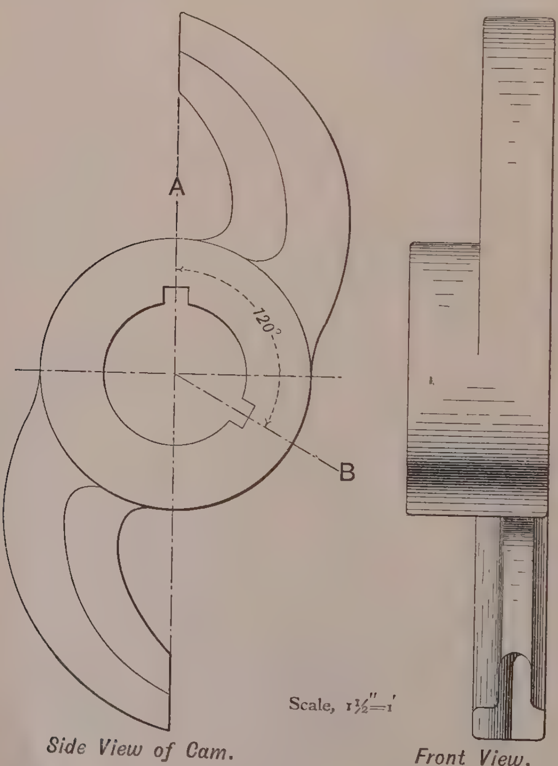
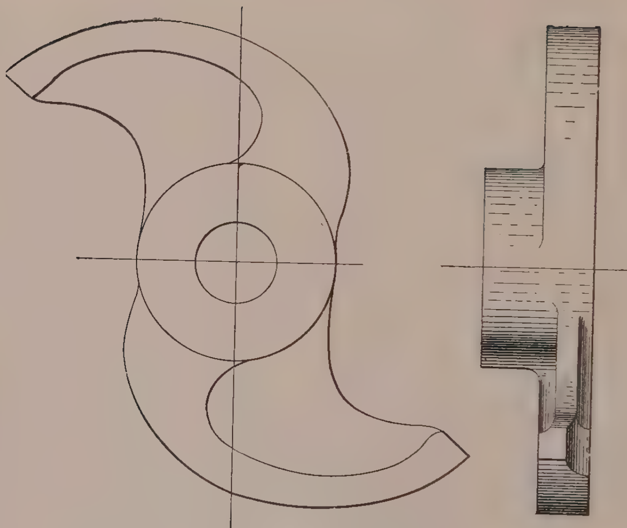


Fig. 36.

armed, but the two-armed cam is practically the only one in use now. The patterns generally adopted for cast-iron and for steel cams are shown in Figs. 35 and 36

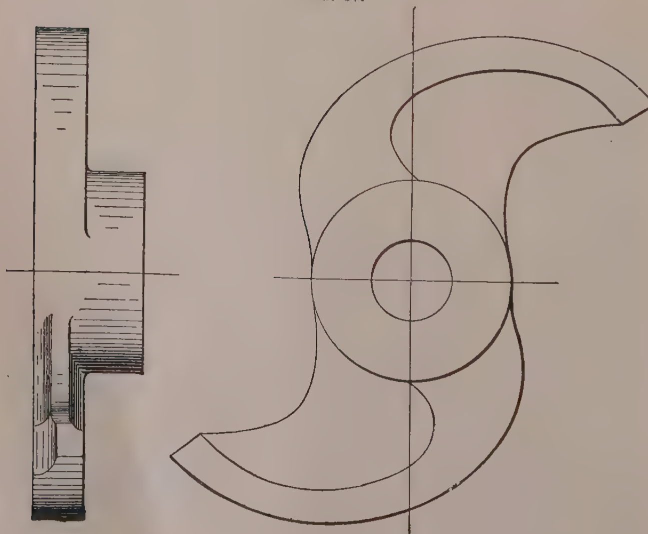
respectively, the departures from these being only slight. Cams may be either right- or left-handed. A right-handed cam is one that runs on the right-hand side of the stamp when the observer is looking in the direction in which the upper arm of the cam is revolving; hence the boss or hub of the cam is also on the right-hand side, whilst in a left-handed cam it is on the left-hand side when looking in the direction of revolution. Views of right- and left-handed cams are shown respectively in Figs. 37 and 38.

Cams are made either of cast-iron or steel. If cast-iron be used it should be of the very best quality, a tough iron, low in phosphorus and of fine uniform grain being selected; the boss is often strengthened by an iron hoop, as shown in Fig. 35. which represents the usual shape and dimensions of a cast-iron cam. The proper material for the cam is, however, cast steel, good open-hearth steel containing about 0·4 per cent. of carbon being a very satisfactory material for the purpose. Chrome steel has also been used for cams, but is no better than ordinary cast steel for this purpose. The delay occasioned by the breaking of a cam is so great, that every possible precaution should be taken to prevent such an accident, which is about the most troublesome one that can happen in a mill. In a very large mill it may be possible to keep a spare cam shaft ready fitted with cams to replace a cam shaft upon which one of the cams may have been broken, but this is not customary, and, in moderate-sized mills, scarcely possible. When a cam breaks, all the stamps run by the shaft upon which it is keyed have to be hung up, and the shaft with its cam and pulley lifted out of its place by heavy tackle, which is no easy matter, seeing that a cam shaft complete



Scale 1″=1′

Fig. 37.



Scale, 1″=1′

Fig. 38.

with cams and pulley for a ten-head mill weighs about two tons. The cams have then to be removed until the broken one can be got at; meanwhile the necessary key-seats will have been cut in a square cam to replace the broken one. The cams have then to be keyed up in their places again, and the shaft hoisted up into its bearings. This work usually takes the best part of a day, even in well-fitted mills. At one time an attempt was made to construct cams in two halves held together by strong stirrups and wedges. It was found, however, that more trouble was entailed by the getting loose and slipping of these sectional cams than was compensated for by the comparative facility of their renewal; hence this system never came into favour, and, with the great strides made of recent years towards the perfection of steel castings, has now been entirely abandoned. Every well-equipped mill should have cast-steel cams, even though iron be used for every other part. Steel cams and cast-iron tappets work very smoothly and wear well together.

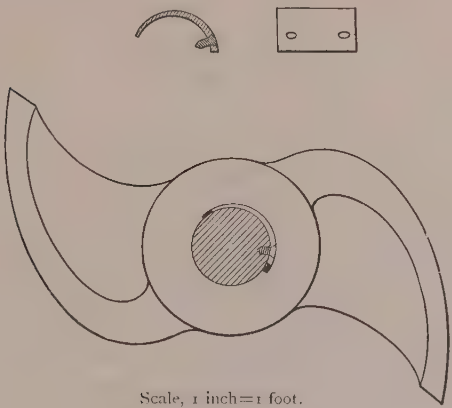
A cast-iron cam needs also to be very much heavier than does a steel one. The strengthening rib of the arm must be deeper and thicker, and the hub also must be of larger size. The diameter of a cast-iron hub must be quite twice that of the cam shaft, whilst in the case of a good steel casting $1\frac{3}{4}$ times the diameter is sufficient, although it is usually made a trifle more. In each case the width of the projecting portion of the boss is made about equal to that of the working face of the cam, the latter being generally from 2 to 3 inches. The face of the cam which comes next to its stamp stem should be carefully planed so as to be quite true and accurately at right angles to the bore of the cam; this precaution is often neglected, in spite of its importance. If the cam is

thus planed, it can safely be set not more than $\frac{1}{32}$ inch away from the stamp stem, and the friction caused by the one-sidedness of the lift is thus reduced to a minimum. The cam must be carefully bored out to a "driving fit," on the cam shaft; as already recommended, a gauge of the exact size should be prepared for this work.

Cams should be fitted with two keys each, the key-ways being 120° apart; some makers use only one key, but two keys are always advisable for stamps of over 700 lbs. at any rate. The keys should be of ordinary dimensions, have a considerable taper, and of course be driven towards the stamp stem and not from it. The whole tendency of the reaction between the stamp tappet and cam is to force the cam away from the centre of the stamp stem, and this tendency needs, of course, to be guarded against.

Within the last few years several methods of attaching the cam to the shaft without the use of key-ways have been introduced. As a key will only hold the cam-boss tightly when driven into it longitudinally, it is necessary that each key-way should be at least twice the length of the key, and almost the only method of attaining this object satisfactorily is by slotting out a continuous key-way for the full length of the cam shaft. It will be seen presently that each cam must be placed at a definite angle to all the others, so that, the position of the key-way in the shaft being fixed, each cam requires to have its key-way cut in the exact position to give this definite angle. This difficulty, which renders a cam once slotted only suitable for one particular place, has been very neatly overcome by the Blanton cam, shown in Fig. 39. All the cams are machined by the maker with a wedge-shaped recess as shown; into this fits loosely a piece of the shape shown in the figure, which is practically a wedge curved to the

radius of the cam shaft, and these wedges are suitably distributed along the shaft, and held in their proper relative positions by means of a couple of small set-screws. Each cam can be slid along the cam shaft and slips easily over the wedge; as soon, however, as the shaft is rotated in the proper direction for driving the mill, the weight of the stamps tightens the cams upon their wedges and holds them firmly in place. When a cam breaks and it is



Scale, 1 inch = 1 foot.

FIG. 39

required to take the cams off the shaft, they can be at once released by a smart blow of a hammer on the under side of the cams, so as to turn them a little in the opposite direction. Obviously, therefore, the laborious operation of renewing a cam is immensely facilitated, and loss of valuable time is prevented. It would seem at first sight as though the shearing strain on the set-screws must be enormous; but in fact the part that these screws really play is merely to keep the wedge from shifting until the

cam has gripped it, the arrangement acting from that moment as a true friction grip, which bites the harder the greater the strain to be resisted. The Blanton cam has proved quite successful in practice, and may be looked upon as one of the leading recent improvements of the stamp-mill; it is made by Messrs. Fraser and Chalmers, Limited. Somewhat similar devices are the Davis patent self-tightening cam-fastening, made at Denver, Colorado, which consists of a short feather, oval in cross-section, which drops into a shallow depression in the shaft, whilst the cam slips over it and tightens against it when the mill is running; and the cam-fastening introduced by the Humboldt Engineering Company of Cologne, Germany, which similarly consists of a short feather, parallel longitudinally, but wedged-shaped in cross-section, the action of which is quite similar. It is obvious that these two last-named fastenings are applications of the same principle as the Blanton grip, but are mechanically inferior; neither of them has yet been employed to any extent. In the most recent modification of the Blanton grip the cam shaft is machined so as to have ten curved taper faces running along its full length; the cams are correspondingly bored out, so as to slide on to the shaft quite easily; they are then tightened up with one or two blows of a hammer, and are kept tight by the weight of the stamps as in the older pattern. This new pattern allows any cam to be placed in any required position on the shaft without the least fitting, and is a very decided improvement.

When the cams are properly secured in their places so as to be unable to shift laterally—a matter of great importance—the reaction between the tappet and the cam tends to displace the entire cam shaft longitudinally

in a direction away from the stamp stem towards the respective cam. Of course this tendency to shifting must be resisted, but it should be clearly borne in mind that there is a tendency to move in this one direction only, so that, if the cam shaft is fitted with collars, only one is needed to prevent the cam shaft shifting laterally away from the stamps. This tendency can be—and is by all good makers—entirely overcome when ten heads of stamps are worked by one cam shaft, by making the cams which drive one set of stamps right-handed and the others left-handed. When only five stamps are worked by one cam shaft, this lateral thrust can be overcome by having two cams of one kind and three of the other; but more usually they are all five either right- or left-handed, the driving wheel being then keyed on the end of the shaft away from the cam hubs, *e.g.*, if the cams are left-handed the pulley or spur wheel should be on the right-hand end of the cam shaft, so that its turned boss may work against the turned end of the cam shaft bearing. A cam known as the "Bally" cam was patented with the object of overcoming this lateral thrust, so arranged that instead of the two arms of each cam working, as is usual, the same stem, they worked adjoining ones. It did certainly prevent lateral thrust; but, as I have already shown, this can be done in other ways, whilst this "Bally" cam had the disadvantage that it did not continuously rotate the stamp stem, but simply turned it backwards and forwards through a certain arc, and thus tended to wear the shoes and dies unequally. It has never really come into use. Another device is that known as the "Hart" cam, in which each cam arm consists of two limbs, one of which works on either side of the stem so as to equalise side thrust, whilst, in order to enable it to revolve the stamp, one

limb is made slightly longer than the other. This system seems as unlikely to be adopted as the preceding one.

Cam Curve.—The proper curvature to be given to the working face of the cam is a matter of paramount importance. Fortunately the setting out of this curve is a very easy matter. I shall here only enter into the technical portion of the subject, leaving the purely geometrical part to be dealt with in a separate appendix (page 565), where proofs will be found of the assertions here advanced. The object of the cam is to convert the uniform rotary motion of the cam shaft into an upward motion of the stamp stem, such that the rate of lifting shall be uniform, the action being intermittent, so that time is given it to admit of its falling freely with uniformly accelerated velocity under the action of gravity. The curve which will convert uniform rotary motion into a uniform lift is one of the involutes to a circle, the radius of this circle being equal to the horizontal distance between the axes of the cam shaft and the stamp stem. It is a property of this involute that the lengths of the arc of this circle traversed by the rotating cam in a given time shall be equal to the amount of vertical lift during the same time. Or, if the amount of lift be called h and the radius of the generating circle r, both in inches, the cam moving through $a°$ during the lift h, then, in the case of the two-armed cam, which will alone be considered here—

$$h = \frac{\pi r a}{180}$$

$$a° = \frac{h}{\pi r} 180°$$

$$r = \frac{180\ h}{\pi a}$$

These equations provide the means of connecting these three important factors. As a general rule, r is fixed by the conditions of construction of the mill; it is equal to the sum of the radii of the stamp stem and cam shaft plus a small amount of from $\frac{1}{8}$ inch to $\frac{3}{16}$ inch for clearance. The value of r being thus fixed, the angular motion of the cam corresponding to a lift of 1 inch will always be $\frac{a^\circ}{h}$ or $\frac{180^\circ}{\pi r}$. From these data it is perfectly easy to set out the cam curve. Describe a circle, Fig. 40, with centre C and radius r inches. Draw any radius CO of the circle, and then set off successive radii at angles equal to $\frac{180^\circ}{\pi r}$ apart, $C1$, $C2$, $C3$, &c.; at the end of each of these radii set off tangents, making $1I$ equal to 1 inch, $2II$ equal to 2 inches, $3III$ equal to 3 inches, and so on. The curve joining the points O, I, II, III, &c., will be the curve required. It is clear that the lowest possible point to which the tappet can descend in practice is fixed by the radius of the hub of the cam, since the tappet must never be allowed to strike the latter. Call this radius k; then the lowest possible position of the tappet will be at a point situated at a height k above the horizontal through C, whilst its highest position will be $k+h$ above it. A length of cam-surface corresponding to a lift of $k+h$ must therefore be set out; the only portion of this curve that ever comes into action is that corresponding to h, the portion of the root of the curve corresponding to k being never required; in fact, this portion may be given any desired shape provided only that it is so designed that no portion of it can come into contact with the tappet. It is a property of this involute that the tangent to it will always be hori-

zontal, when the line of lift is vertical. Hence, by adopting this shape, the whole of the rotative force of the cam

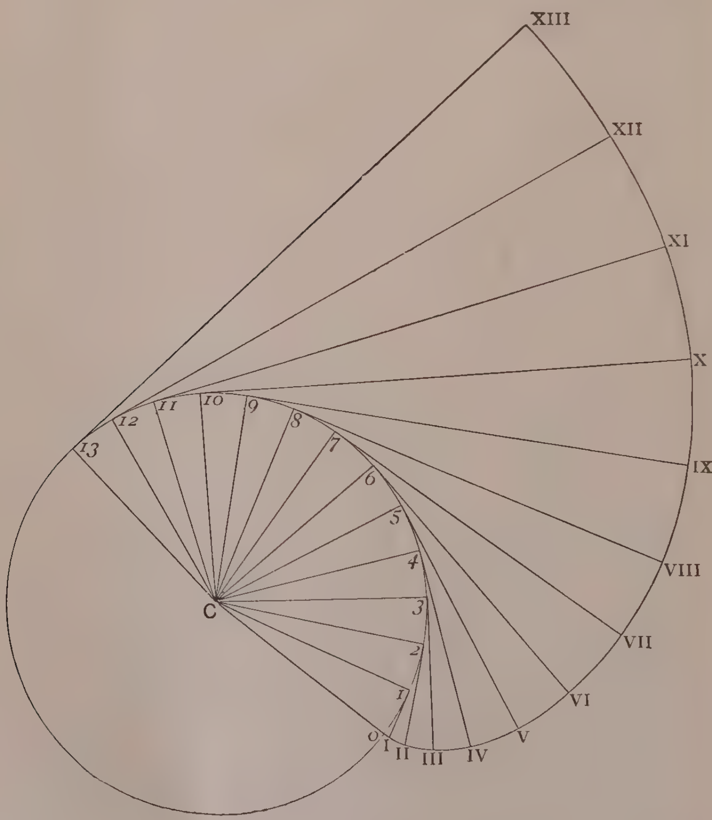


FIG. 40.

is converted into vertical upthrust (disregarding friction for the time being). If the cam were to be made a true

involute throughout the portion of the curve corresponding to h, the stamp would be lifted with uniform velocity, and on leaving the cam it would still be moving upwards with the same velocity, so that the lift actually given would be really somewhat greater than that due to the cam itself. To obviate this, so much of the cam curve as corresponds to the top quarter of an inch of lift is shaved off, and the curve so modified at the point is the curve actually used for the cam; the motion of the stamp produced by the curve so modified is practically the same as would be produced theoretically by a point moving on a true involute, and may be so regarded.

As an illustration the cam curve required for a 900 lb. stamp of very customary dimensions is here (Fig. 41) drawn on a scale of 3 inches to 1 foot; the lift is taken at 7 inches; the stamp stem is $3\frac{1}{2}$ inches in diameter, and the cam shaft 5 inches; then, allowing $\frac{1}{8}$ inch for clearance—

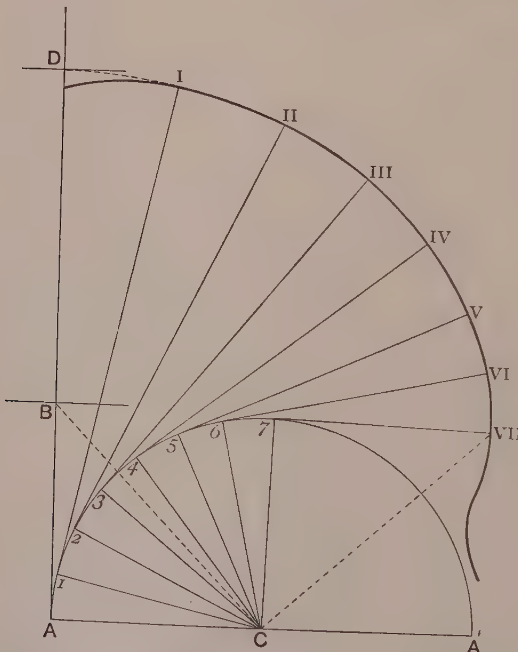
$$r = \frac{3''\cdot5}{2} + \frac{5''}{2} + 0''\cdot125 = 4''\cdot375,$$

$$a^\circ = \frac{7}{\pi \times 4\cdot375} \times 180^\circ = 91^\circ 42'.$$

Angle corresponding to 1 inch of lift $= \dfrac{180^\circ}{\pi \times 4\cdot375} = 13^\circ\cdot6'$.

It is assumed that the cam is to be of steel, having a boss of 9 inches diameter, the value of k being accordingly 4·5 inches. The method of setting out the curve is as follows: Describe the generating circle with centre C and radius $r = 4''\cdot375$. Draw the horizontal diameter ACA' and set out from A the perpendicular tangent AD. On this set off a height $AB = k = 4''\cdot5$, and a height $BD = h = 7''$. The horizontal line drawn through B is then evidently the lowest possible position of the tappet,

and that through D is the highest. From CA are set off successive radii $C1$, $C2$, $C3$, . . . $C7$, each at an angle of 13°6′ to the one next before it, that is to say the angles $AC1$, $1C2$, $2C3$, . . . $6C7$, must each be made equal to



Scale $3'' = 1$ ft.

FIG. 41.

13° 6′. Then draw tangents $1I$, $2II$, $3III$, . . . $7VII$ from each of the points 1, 2, 3 . . . 7, equal to $k+h-1$, $k+h-2$, . . . $k+h-h(=k)$, that is to say to 10″·5, 9″·5, 8″· 5. . . 4″·5 respectively. Join the points D, I, II, . . . VII, &c., and from VII continue the curve in any form that will

give the desired shape for the cam arm, so as to combine sufficient strength with lightness till it reaches the boss. Take a point a quarter of an inch below D, and from that draw in the curve by hand as shown until the curve of the finished cam represented by the black line is obtained, the dotted line showing the true involute.

The construction of one cam arm only is here shown, the other being quite symmetrical to it. There are various other ways of drawing the involute to a circle, but the one here given is the easiest and most accurate. If preferred, points may be obtained half an inch instead of an inch apart by making each angle one-half of the above amount, namely, $\frac{90°}{\pi r}$, and by decreasing the successive tangents by half an inch instead of by one inch.

In practice the curve should be thus set out in full size on a sheet of stout drawing paper, pasted on a piece of well-planed board, in which the curve can be accurately cut out when it has been drawn, and metal templates then made from it.

Care must be taken that the point of the cam as seen in plan is cut diagonally to a curve which must be an arc of the circumference of the tappet, so as to allow the whole tappet to clear the cam and to fall at once, immediately the cam has passed the centre line of the stamp. The fillet thus formed is well seen in Fig. 37.

It is obvious that the above equation, $h = \frac{\pi r a}{180}$, indicates the limit of the amount of lift by a two-armed cam for any given radius. However slowly the cam be made to revolve, it is impossible for $a°$ ever to be quite equal to 180°, seeing that the stamp must take some definite time in falling, and that the angular speed corresponding

to this time must always be something, however small; therefore the maximum possible value of h is less than πr. When higher lifts are required, a single-armed cam must be used; in this case a° become 360°, and the limiting value of h becomes $2\pi r$. Whenever possible, single-armed cams are avoided, because they limit the speed of the mill and moreover cause greater loss of power than do the double-armed cams, seeing that the cam shaft has to revolve twice as fast and that a longer cam-surface is needed for the same speed and lift; moreover, the cam shaft is not so well balanced. In the rare cases when it is necessary to produce a lift still greater than this, some form of curve other than the normal involute would have to be used, or else r would have to be increased by modifying the arrangement of the mill.

Arrangement of Stamps.—It has hitherto been taken for granted that five stamps always compose one battery, and five is indeed the customary number in a very great majority of instances, although batteries of from two to six stamps, and occasionally even more, have been built. It cannot be proved mathematically that five is the absolutely best possible number, but it will be seen from the following considerations that it is the most convenient. Less than three stamps give a very small mortar box, and entail a large number of wearing parts, besides multiplying battery frames, cam shafts, pulleys, &c., whilst more than six stamps would require an inconveniently large and heavy mortar. The end stamps of a battery are always less effective than the central ones, for the reason that the former have a screen surface on either side of them, and the latter on one side only; for this reason batteries of two or three heads are proportionately less effective than the larger ones. On the other hand, it

is just as bad to have too many as too few stamps worked by the same cam shaft, as any injury to one head may cause the stoppage of all the stamps on that shaft. It is found that an odd number of stamps works better than an even number in the mortar, giving a more uniform wash of the pulp and providing a central stamp which will serve to control the action of the battery. Five has accordingly been fixed upon from the result of long experience as the most convenient number of stamps to work in one battery, small portable and prospecting mills having usually three stamps; the five-stamp battery is accordingly looked upon as the standard size.

It is not by any means a matter of indifference in what order the stamps of a battery are allowed to fall; the main objects to be attained are uniform working, which shall not admit of the accumulation of crushed quartz in either end of the mortar box, and a steady uniform wash backwards and forwards of the pulp within the battery with as little splashing as possible. Two general principles have been enunciated by mill men, compliance with either of which gives satisfactory results, although these principles are to some extent antagonistic ones. These are: (1) that neighbouring stamps shall never be allowed to fall in succession, and (2) that while any given stamp is falling, its neighbours shall be rising. Numbering the stamps successively 1, 2, 3, 4, 5, beginning at the driving end of the shaft the first principle is complied with by making the succession of falling 1, 4, 2, 5, 3, or 1, 3, 5, 2, 4, which two orders are, it will be seen, the same except that one runs first from the centre towards the right hand, and the other from the centre towards the left hand. Compliance with the second principle demands the orders 1, 5, 2, 4, 3, and 1, 4, 2, 3, 5, which bear to

each other the same relation as do the first pair. One or other of these four systems is now usually adopted, but the orders 1, 5, 3, 4, 2 and 1, 3, 2, 4, 5, are also sometimes used. These latter are not to be recommended, but any of the four first named give about equally good results and can be used with confidence.

The action of the stamp is twofold—namely, crushing the ore in the first place, and afterwards expelling the pulp, which consists of crushed ore suspended in water, through the screen apertures by its piston-like action. As regards crushing power, all the stamps in a battery box are equally effective, but as regards their expulsive effect the end stamps do a lesser duty than the central ones, because part of their propulsive power is wasted against the dead ends of the box. There is accordingly a tendency for crushed stone to accumulate under the two end stamps, and if the mortar box and the line of discharge are not accurately horizontal, there will be a tendency for it to accumulate under one or other of the end stamps only. This latter defect can evidently be obviated by taking great care that the horizontality of all parts of the mortar is rigorously maintained, and the former is usually met by giving the end stamps a slightly longer drop (say $\frac{1}{4}$ inch to $\frac{1}{2}$ inch) than the middle ones. Inattention to the due regulation of the water supply is also a frequent cause of similar irregularities in the working of the mill. When a cam shaft works ten stamps, corresponding stamps of the two batteries should fall successively. Thus, if the second-named order be adopted, the order for ten stamps will be 1, 6, 5, 10, 8, 7, 4, 9, 3, 1, and so on for the other orders. The cams must be set so as to produce the desired order of succession, their key-ways being suitably cut by the makers,

or, in case of Blanton cams, the holes for the set-screws drilled in their proper places on the shaft. Care must be taken that their distribution round the cam shaft shall be quite uniform, and this is easily done by making the angle between each two successive (not neighbouring) cams equal to 180° divided by the number of cams on the cam shaft. This formula applies only to two-armed cams, since these give one complete drop for an angular motion of the cam shaft of 180°. Thus, if there be ten such cams on one shaft, the angular distance between successive cams must be $\frac{180°}{10} = 18°$. When spare cams are supplied, their key-ways are never cut, as it is obviously quite impossible to say which cam is likely to be the first to require replacing. In a big mill spare cams are key-seated as required by means of a slotting machine, but in smaller establishments they have to be cut by hand by means of strong flogging chisels driven by a heavy hammer, a reamer being usually driven down afterwards to secure accuracy of fit. It is a good plan to have at hand a block consisting of a short section of cast-iron or steel of the same depth as the cam-boss, accurately turned to the same gauge as the cam shaft, and slotted to correspond accurately to the key-ways of the shaft. This greatly facilitates fitting the keys. The position of one of the key-seats of any one given cam ought to be accurately known by its distance from the line joining the points of the cam arms, and knowing this, it is easy to calculate the angular distance from it of the key-seats of any of the other cams; hence as soon as a breakage occurs, the engineer in charge can at once commence cutting the key-seats in a spare cam, so as to get it ready for replacing the broken one. This

work is, of course, all obviated by using cams on the Blanton principle.

The rotative power of the cam shaft is exerted in various ways besides its normal application to lifting the stamp vertically. In making this vertical lift it has not only to overcome the action of gravity but also that of the friction of the stamp in its guides. During the lift the stamp is suspended upon the arm of the cam, and as the point of suspension is to one side of the axial line of the stamp in which the centre of gravity of the latter lies, there is a tendency on the part of the stamp to assume such an inclined position as will bring its centre of gravity vertically beneath its point of suspension, as shown diagrammatically in Fig. 42, where this action is of course greatly exaggerated by exaggerating the distance between the cam and the stamp stem. In this diagram C is the cam in cross-section, T is the tappet, and G the centre of gravity of the stem, which assumes a position in the vertical line XG as shown.

Fig. 42.

This action causes a lateral thrust on the guides, and at the same time causes a lateral reaction between the tappet and the cam, tending to force the latter away from the stamp. It can be diminished by bringing the cam as close as possible to the axis of the stamp and by removing the centre of gravity of the stamp as low down as possible, which effect will be obtained by increasing the weight of the head relatively to that of the other portions of the stamp.

The cam also exerts an action on the tappet that tends to revolve the entire stamp round its axis in the direction in which the cam itself is moving, owing to the friction between the cam and the tappet. The amount of this friction depends upon the velocity of the cam, the weight of the stamp, and principally on the nature and lubrication of the two surfaces in contact. Since the revolutions of the tappet and cam take place round axes perpendicular to each other, whilst the contact of the revolving surfaces takes place in a straight line parallel to the axis of revolution of the cam, it follows that every point on this line of contact on the cam surface is moving at a uniform rate, whilst on the tappet surface every point is moving with a varying velocity, depending on the distance of the point from the axis of revolution. There must therefore be rubbing and not rolling friction between these surfaces, and lubricants must be employed to diminish this friction as far as possible. The net result of the friction is, as already stated, to rotate the entire stamp about its axis, and this circular motion has the great advantage that the wearing surfaces, particularly those of the shoe and die, are affected uniformly, thus contributing greatly to the regularity of working of the entire machine. Much nonsense has been written to the effect that this whirling movement has a grinding action on the quartz between the stamp and die, but no one who has watched a stamp-mill closely will need to be told that this is not the case. The rotation of the stamp takes place during the lift, and continues very slightly during the commencement of the descent, being rapidly neutralised by the friction of the stamp in the guides, so that it has entirely ceased by the time the shoe strikes the quartz. It is easy enough to prove this by taking a

diagram by simply holding a piece of chalk steadily against a stamp stem whilst working. It will then be seen that the chalk traces a steep spiral during the ascent of the stamp, but that during the descent it makes an almost vertical line. In practice the stamp should never be allowed to rotate through more than 30° at each stroke, so that it should make one complete revolution in never less than twelve drops. If it revolves faster than this, it is merely wasting power that could be better employed in driving the mill, as the above speed of rotation is ample to secure uniformity of wear.

Lubrication.—In order to economise power and to protect the cam and tappet faces from undue wear, it is necessary that they should be thoroughly lubricated, any deficiency in this respect at once making itself known by the stamps spinning round too rapidly. Many different lubricants have been used for this purpose. Some use anti-friction grease, others merely tallow; a composition of tar, beeswax, and resin, boiled together, is sometimes used. Some makers of mining machinery sell a special composition of this character as a cam lubricant. It is best, however, to use some material quite free from grease, so that if any of it finds its way into the mortar box or on to the plates, the process of amalgamation is not interfered with. Common molasses may be and often are used, but the best lubricant for the cam is soft soap thickened with a small quantity of finely-ground graphite. A pot of this mixture should be kept on the working platform, and whenever a stamp shows the least tendency towards too rapid revolution, a little of it should be rubbed on the face of the cam by means of a stick kept for the purpose. Many mill men fasten strips of canvas or oilcloth (cut so as to allow the

stamp stems to pass through) below the lower guides, for the purpose of catching any portions of the lubricant that may become detached and which might fall into the mortar were it not for this shield.

Height of Drop.—The question as to what is the right length of drop to be given to the stamps, and the closely connected one as to what is the best speed at which to run, have of recent years undergone a marked modification with the introduction of heavier stamps. There is a pretty general consensus of opinion that heavy stamps, high speeds, and short drops are the conditions that produce the most economical results in crushing, although in some districts, *e.g.* in Colorado, a different practice prevails owing to reasons that will appear later on (see page 467).

With 900 lb. stamps a six-inch blow is sufficient to crush the firmest piece of quartz completely, and in order to get the maximum effect out of a mill it must, of course, be run at the highest possible speed. This maximum speed depends upon the height of drop and the distance between the axes of the stamp stem and cam shaft.

It has already been stated that this distance, the height of lift and the angular movement of the cam are connected by the equation $h = \frac{\pi r a}{180}$.

Let the cam be so constructed as to give N drops per second; the cam shaft will then be making $30N$ revolutions per minute. Also—

$$\text{Time of one complete drop} + \text{lift in seconds} = \frac{1}{N}.$$

Now let H be the height of the lift $\left(= \frac{h}{12}\right)$, and

$R\left(=\frac{r}{12}\right)$ the distance between the centres of the stamp stem and cam shaft, both being expressed in feet. If we suppose that the stamp has no interval of rest, then will the time of one drop + time of one lift oocupy one semi-revolution of the cam shaft; obviously this will be the maximum speed at which the stamp can be driven. Now the time occupied by a body in falling *in vacuo* from the height H is $\sqrt{\frac{2\,H}{g}}$ seconds, where g is the accelerating action of gravity (about 32·2 feet per second).

It follows from the properties of the involute that the portion of a semi-revolution of the cam shaft occupied in lifting the tappet is $\frac{H}{\pi R}$. The time occupied by the lift is accordingly $\frac{H}{\pi R} \cdot \frac{1}{N}$ and

$$\frac{1}{N} - \frac{1}{N}\left(\frac{H}{\pi R}\right) = \sqrt{\frac{2\,H}{g}}$$

$$N = \sqrt{\frac{g}{2\,H}}\left(\frac{\pi R - H}{\pi R}\right)$$

which formula gives the maximum possible number of drops per minute for a given construction of battery and given depth of drop, supposing the stamp to be working *in vacuo* without friction, and to have no interval of rest at all. Friction must, however, come into play, and even the latter condition can never be realised in practice, as the stamp must always be allowed to remain at rest for a certain time, because in actual work the stamp, after it strikes the die, rebounds slightly, and then falls again. If the cam were to meet the tappet before the stamp had come to rest for the second time, the shock would be

violent enough to do serious damage, most probably to break off the arm of the cam. Such a collision, generally spoken of as "camming," must accordingly never be risked.

The minimum interval of rest which can safely be allowed in a stamp-mill is one-tenth of a second. When this interval of rest is adopted, the above formula would become modified thus:—

$$\frac{1}{N} = \frac{1}{N}\left(\frac{H}{\pi R}\right) + \sqrt{\frac{2H}{g}} + \frac{1}{10}$$

$$N = \frac{10\sqrt{g}(\pi R - H)}{\pi R(10\sqrt{2H} + \sqrt{g})}$$

When $r = 4{\cdot}375$ inches and $h = 7$ inches ($R = 0{\cdot}365$ ft., $H = 0{\cdot}583$ ft.), the above formula makes N equal to 1·69, or the maximum possible number of drops per minute would be $1{\cdot}69 \times 60 = 101$, still neglecting friction.

I have quite recently shown[1] what is the actual rate of falling of an ordinary stamp under normal conditions, and that for all practical purposes it can be expressed by substituting in the theoretical equation $T = \sqrt{\frac{2H}{g}}$, a coefficient f for g, the mean value of f being 27·5, so that $\sqrt{f} = 5{\cdot}24$. By substituting this value for $\sqrt{g}$ in the above equations, a close approximation to practical results will be obtained. For example, if the last equation be written thus:—

$$N' = \frac{52{\cdot}4\ (\pi R - H)}{\pi R(10\sqrt{2H} + 5{\cdot}24)}$$

and the same values be again assumed for r and h, N' will be found to be equal to 1·60, or the maximum number of drops per minute would be 96. In practically

[1] *Stamp-Mill Indicator Diagrams*, Amer. Inst. Min. Eng., 1898.

running a stamp-mill, the dimensions of which corresponded very closely with the figures here assumed, I found that camming was just avoided at a speed a very little above 95 drops per minute.

In practice the height of drop is usually between 5 and 12 inches, and the following table shows the length of time occupied respectively by a body falling *in vacuo*, and by a stamp falling in normal mill practice from the various heights between these extremes, in decimals of a second:—

Height of drop.	Time of falling.	
	Theoretical.	Actual.
5 inches	0·161 second	0·174 second
6 ,,	0·176 ,,	0·190 ,,
7 ,,	0·190 ,,	0·206 ,,
8 ,,	0·204 ,,	0·220 ,,
9 ,,	0·216 ,,	0·233 ,,
10 ,,	0·227 ,,	0·245 ,,
11 ,,	0·239 ,,	0·258 ,,
12 ,,	0·249 ,,	0·268 ,,

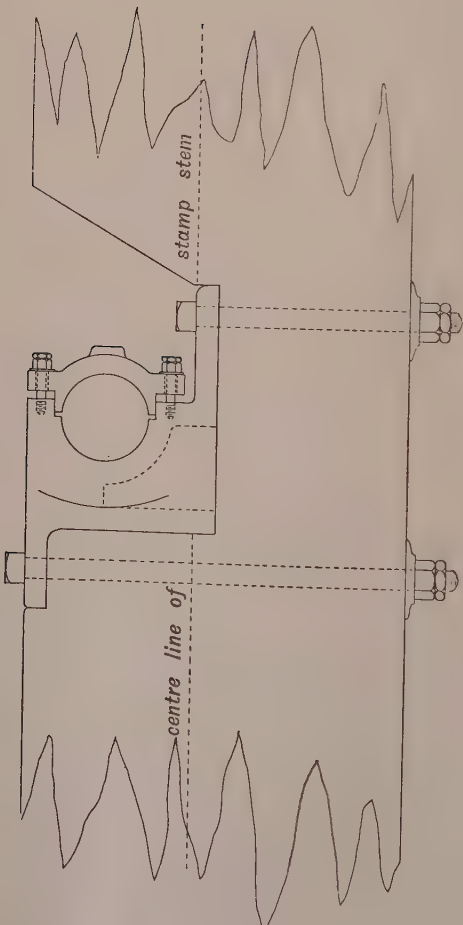
Crushing Power.—Disregarding friction for the moment, the power required to lift a stamp of weight W through the height H is WH foot-pounds.

The crushing effect of a stamp appears to depend mainly upon its momentum, this being the product of its mass into its velocity.

The momentum of a stamp of weight W, acquired by falling from a height H, is $W\sqrt{2gH}$, when friction is disregarded, or $W\sqrt{2fH}$, allowing for friction. Therefore the power required to lift a stamp varies directly as the height to which it is lifted, whilst the effective force developed by it varies only as the square root of that

height. The maximum effect is accordingly obtained from a given number of foot-pounds employed as a lifting power, when W is a maximum and H is a minimum; in other words, the most economical way of employing power in a stamp-mill is by making the weight of the stamp as great and the height of the drop as small as is consistent with convenience in practice, provided that the momentum is still sufficient to both shatter the rock and to propel the shattered particles through the screen. This is also evident from a comparison of the theoretical and actual velocities of falling, the difference between these, or in other words the momentum absorbed by friction, being greater for long drops than for short ones. Furthermore, a short drop allows the number of drops per minute to be increased, and such increase in the rate of falling increases greatly the efficiency of the mill. Experiments by Messrs. Morison and Bremner have shown that the rate of crushing is practically proportional to the speed, the rate of crushing increasing, however, a little more rapidly than the speed when the number of drops is between about 75 and say 110 per minute. Their experiments also show that the crushing power increases more rapidly than the weights of the stamps up to a limit of about 1,400 lbs. It is worthy of note that these are the practical conclusions to which modern experience is decidedly tending; stamps of 1,200 lbs. and 1,250 lbs. weight are coming into use; a 1,400 lb. stamp was even tried in California, but was not found satisfactory.

Cam Shaft Bearings.—The proper shape of the cam shaft bearings depends upon that of the mill frame and on the direction and manner in which power is transmitted to the shaft. A usual form is shown in Figs. 43 to 45, which will be found thoroughly satisfactory in practice.

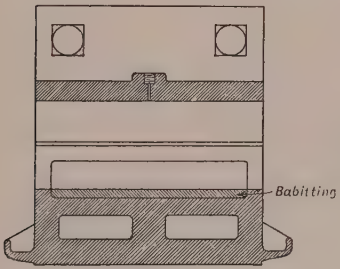


Side Elevation

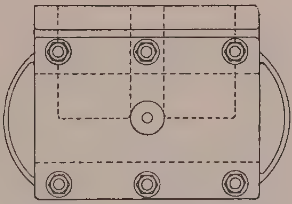
Scale, $1'' = 1'$

Fig. 43.

If the cam shaft be driven by a belt, the pull of which is downwards, or at any rate not decidedly upwards, as is mostly the case, the lower bearing will take the whole of the work, as it is obvious that the entire weight of the cam shaft together with that of the stamps acts vertically downwards. Many makers, in fact, dispense with a cap altogether, and prefer to let the shaft run in the bearing with its upper half quite unprotected, on the ground that it is easier to keep the shaft clean and in good running order and thoroughly lubricated than when it is covered



Scale, 1″ = 1′
Transverse Section.
FIG. 44.



Plan
Scale, 1″ = 1′
FIG. 45.

up. Against this view it may be said that there is usually some quartzose dust flying about a stamp-mill, which would be apt to work into and cut the uncovered part of the shaft journal; nevertheless open bearings are largely used and are found quite satisfactory under ordinary conditions. It is at any rate certain that, when the bearings have caps, these should be, though light, very well fitting and supplied with good automatic needle lubricators; the bolts that hold the cap need only be light ones. Diagonal bearings are sometimes used; their employment is only permissible when the belt

exercises a diagonal pull, so as to press the shaft against the bearing. It is very rare indeed that the cap of a cam shaft bearing has to take any portion of the strain.

The best material for the actual cam shaft bearings is Babbitt metal; brass is sometimes used, but is less suitable. Plain cast-iron boxes may be used where economy in first cost is an object.

The rate of revolution of a cam shaft is, of course, slow; but the great weight of the shaft with its cams, in addition to that of the stamps, is apt to cause a good deal of friction, and great attention should accordingly be paid to the lubrication. All cam shaft bearings should have a drip-pan, as shown in the figures, cast on the base of the bearing, so as to hold a good deal of oil, and prevent this finding its way down the side of the battery posts. Should the bearings not have a drip-pan cast on, a pan made of sheet-iron and completely surrounding the base should be set below each bearing, so as to catch all superfluous lubricating material. It should be the duty of one of the men to carefully and thoroughly clean out these drip-pans at stated times, at least once in twenty-four hours.

Power Required.—The power applied to the cam shaft pulley is absorbed in the following actions:—

1. In the friction of the cam shaft on its bearings.
2. In the friction of the cam against the tappets.
3. In the friction of the stamp stem against the guides.
4. In the dead lift of the stamp.

1. In calculating this item, it must be remembered that the friction of the cam shaft depends on the weight that presses this shaft upon its bearings. This weight

consists of the weight of the shaft itself, the weight of the driving pulley or wheel, the weight of the cams, and the weight of the total number of stamps continuously being lifted at the same time. To calculate the number of stamps being supported at any one time, the formula $a^\circ = \frac{h}{\pi r}180^\circ$ has again to be applied. From this it is clear that in one complete revolution of the cam shaft, each stamp is supported for a period corresponding to $\frac{2a^\circ}{360^\circ}\left(=\frac{h}{\pi r}\right)$. Hence the average number of stamps carried at any one time is equal to $\frac{hS}{\pi r}$ where S represents the total number of stamps driven by the cam shaft. This is the number of stamps supported simultaneously, and although these stamps are acting on the cam shaft at different angles and lengths of leverage, yet their total thrust on the bearings will be the sum of their weights. In some cases the pull of the belt will also have to be added, especially when this is vertically downwards. The accepted formula for power so absorbed is :—

$$\text{Foot-pounds of work absorbed in one revolution} = \frac{\text{Total resultant weight} \times \text{diameter of shaft in inches.}}{90}$$

From this the horse power is easily ascertained. To take a concrete example, say a battery of ten heads of 900 lb. stamps making 90 7-inch drops per minute, the construction of the cams being that already given (page 205). In this case the expression $\frac{hS}{\pi r}$ is equal to 5·1 nearly, and we shall then have for the pressure on the cam shaft bearings :—

Weight of cam shaft 15 feet long, 5 inches in diameter . .	1000 lbs.
Weight of pulley	2050 ,,
Weight of 10 cams, each 141 lbs.	1410 ,,
Weight of 5·1 stamps, each 900 lbs.	4590 ,,
Resultant pull of belt, say about	900 ,,
Total pressure	9950 lbs.

Foot-pounds of work absorbed per revolution :—

$$\frac{9950 \times 5}{90} = 553.$$

Since the cam shaft makes 45 revolutions per minute,

$$\text{H.P. absorbed} = \frac{553 \times 45}{33{,}000} = 0{\cdot}754 \quad . \quad . \quad . \ (1)$$

2. The cam curve being always parallel to the horizontal face of the tappet at the point of contact, there is no lateral thrust developed beyond that due to the friction. The friction may therefore be taken as that due to dragging a plane surface of length equal to the length of the cam curve underneath a weight equal to that of the stamp, the coefficient of friction being that due to the nature and lubrication of the surfaces of the cam and tappet. This coefficient may fairly be taken at 0·08; it has already been pointed out that this is not a case of true rolling friction, seeing that the two surfaces in contact are moving at different velocities. The friction between the cam and the tappet would evidently be a maximum were the stamp so held in the guides as to be incapable of revolving; in that case the friction in the guides due to the motion of revolution would be *nil*. If, on the other hand, the tappet were to move (if possible) at the same rate as the cam, the friction at this point would be *nil*, whilst that in the guides due to the revolution of the stamp stem would be

a maximum. In other words, the effect of the rotation of the stamp is to transfer a portion of the friction developed between the cam and tappet to the surface of the stamp stem in contact with the guides. The same reasoning applies to the slight tendency towards a motion of translation in addition to that of rotation, generated in the stamp by the movement of the cam. These are both very small amounts; when a mill is in good running order, the grip of the hand is sufficient to keep the stamp from revolving. There will therefore be no perceptible error if the maximum amount of friction possible, namely, that between cam and tappet, is calculated, and considered to include the friction due to the rotation of the stamp in the guides. The length of the cam curve can be shown to be $\frac{2kh+h^2}{2r}$ where k is, as before, the height of the starting-point of the tappet, h the height of lift, and r the horizontal distance between the axes of the stamp stem and cam shaft, and the power absorbed by friction is therefore—

$$\frac{\frac{2kh+h^2}{2r}}{12} \times \mu W \text{ foot-pounds.}$$

For the case under consideration the expression for the length of the cam curve becomes—

$$\frac{2 \times 4{\cdot}5 \times 7 + 7^2}{2 \times 4{\cdot}375} = 12{\cdot}8 \text{ inches} = 1{\cdot}067 \text{ feet.}$$

Whence, foot-pounds absorbed in each lift of one stamp

$$= 1{\cdot}067 \times 900 \times 0{\cdot}08 = 76{\cdot}8.$$

At the rate of 90 drops per minute,

$$\text{H.P. absorbed} = \frac{76{\cdot}8 \times 90}{33{,}000} = 0{\cdot}209 \quad . \quad . \quad (2)$$

3. The friction in the guides due to the rotation of

the stamp having been eliminated and included in (2), there remains now to consider the friction caused by the action of the guides in maintaining the stamp in its vertical position. It was pointed out (page 212) that the action of the cam being necessarily on one side of the axis of the stamp stem, this latter tends to assume a position inclined to the vertical at a small angle. There is thus a couple formed, tending to turn the stamp into this position round its centre of gravity, and it is the moment of this force that has to be resisted by the guides. The angle which this position of repose makes with the vertical is a very small one; in the case of a typical stamp, the centre of gravity is about 4 ft. 8 in. below the tappet, and this angle becomes nearly 2°.

The horizontal thrust will accordingly be—

$$Sin\ 2° \times W = 0{\cdot}0333 \times 900 \text{ lbs.} = 30\text{lbs.}$$

This angle is so small and varies so little for the various forms of stamp as now constructed, that it may be looked upon as constant for them all, and the horizontal thrust always taken at $\frac{1}{30}$ of the weight of the stamp. Taking the coefficient of friction at 0·1, the power absorbed in each lift will accordingly be—

$$\frac{W}{30} \times H \times 0{\cdot}1 = 30 \times \frac{7}{12} \times 0{\cdot}1 = 1{\cdot}75 \text{ foot-pounds.}$$

And at 90 drops per minute,

$$\text{H.P. absorbed} = \frac{1{\cdot}75 \times 90}{33{,}000} = 0{\cdot}005 \quad . \quad . \quad (3)$$

4. The power required to lift the weight of the stamp alone we already know to be $\frac{W\ .\ h\ .\ n}{12 \times 33{,}000}$ H.P., where W is as before the weight of the stamp in pounds, h the height of lift in inches, and n the number of drops per minute. This formula gives in the present case

$$\frac{900 \times 7 \times 90}{12 \times 33{,}000} = 1{\cdot}432 \text{ H.P.} \quad . \quad . \quad . \quad . \quad . \quad (4)$$

Collecting the three last-named items, we find that the horse-power required to lift one stamp is—

$$0{\cdot}209 + 0{\cdot}005 + 1{\cdot}432 = 1{\cdot}646 \text{ H.P.}$$

Accordingly that required for ten stamps will be 16·46. Adding to this amount that of the friction of the cam shaft in its bearings, we find that it will be necessary to communicate a force of 17·22 H.P. to the belt of a ten-head mill of 900 lb. stamps designed to make 90 7-inch drops per minute. It will hardly be necessary to insist on the great importance of these calculations in practice, so that the engineer can tell exactly what the indicated power of his motor should be in order to drive his mill under any required conditions. Some margin for safety must, of course, always be allowed, but it will be found that the above calculations give an ample one.

It may be useful to group all the above items into one general formula, calling the number of stamps driven off one cam shaft S, the weight of the cam shaft with cams and pulley M, and its diameter d; we have already seen that the diameter of the cam shaft bears a constant relation to that of the cam hub, depending on the material of which the cam is made. In the case now under consideration—

$$2k = 1{\cdot}8d, \text{ or } d = \frac{k}{0{\cdot}9}.$$

We now get as the formula for work absorbed (in horse-power), taking the coefficients of friction above given—

$$\frac{\dfrac{S\,n\,W\,h}{12} + \dfrac{0{\cdot}1\,(S\,n\,W\,h)}{360} + 0{\cdot}08\left\{ S\,n\,W\left(\dfrac{2\,k\,h + h^2}{24\,r}\right)\right\} + \dfrac{\left(M + \left(\dfrac{h\,S}{\pi\,r}\right)W\right)d\,n}{180}}{33{,}000}$$

Some very important considerations can be deduced from the above formulas:—

1. When gearing takes the place of belting, the thrust against the bearings of the cam shaft becomes horizontal and the resultant thrust is somewhat diminished: but as the total amount of power lost from this source is very small, owing to the slow rate of revolution of the cam shaft, the economy so introduced may in practice be disregarded.

2. It will be seen that the loss of power due to friction is directly proportional to the length of the cam curve engaged in lifting the stamp. The formula for the length of the curve was given as $\frac{2kh+h^2}{2r}$ or $\frac{h(2k+h)}{2r}$, and this is evidently a minimum for a given height of lift when k is 0, and increases as k increases; in other words, the lower down that the cam and tappet engage, the less power is wasted, the loss being least when the action commences in the horizontal line through the axis of the cam shaft. In practice this last condition can never be realised, but the best results are obtained when it is approached as nearly as possible. The lowest position of the tappet being determined by the diameter of the hub of the cam, it is now clear how a steel tappet, which can be made with a smaller hub than a cast-iron one, causes an economy of power in driving the mill. It is curious to note that in this respect the old Saxon stamp-mill is a better machine mechanically than the Californian, as in the former the above condition of $k=0$ is realised; this advantage is, however, far more than counterbalanced by attendant disadvantages.

For the same values of h and k the expression $\frac{2kh+h^2}{2r}$

decreases as r increases, so that force would be economised by keeping the axes of the cam shaft and stamp stem as far apart as possible. This construction would however tend so greatly to increase the weight of the cams and the strain on the cam shaft, that economy of power cannot be obtained by diminishing this factor without making the whole machine unwieldy and less effective in other ways.

3. The calculations for the friction between the stamp stem and the guides only apply when these latter are properly adjusted to the work required of them. It would be possible to so tighten up the guide-blocks as to jam the stem and to increase the power required to lift the stamp indefinitely; but it is evident that any positive pressure exerted by the guides upon the stem will merely result in loss of power as well as injury to the machine, their province being simply to maintain the stamp in a vertical position during its lift, and nothing more.

CHAPTER VIII

FRAMES—GUIDES—HOISTING GEAR—WATER SUPPLY—BINS —GENERAL ARRANGEMENT—ORE-FEEDERS

Mill Framing.—The object of the framing of the mill is primarily to carry the cam shaft, and secondly the guides in which the stamp stems work; these simple duties are complicated by the great rigidity of structure required to withstand the constant jar of the stamps and the powerful pull of the belt or thrust of the gearing. The main portion of the frame consists of the battery uprights which carry the cam shaft bearings. When one cam shaft operates ten stamps it has three bearings, and there are then accordingly three uprights to a battery of ten stamps. When a cam shaft only drives five stamps there are often two uprights to each five-stamp battery, though here too the arrangement of three uprights to a ten-stamp battery may be adopted, the centre upright carrying bearings which support the inner ends of the two short cam shafts. These uprights are always connected by the upper and lower guide beams which are fastened to them. The uprights are usually supported on horizontal battery sills, and are strengthened laterally against the pull of the belt by struts or framing. Incidentally, too, these

uprights usually carry the working platform upon which the mill man stands to attend to the cams, tappets, bearings, &c.

The style of framing to be adopted will depend largely upon the method of transmitting power to the cam shaft, and the position which the lay shaft has to occupy when belting and pulleys are employed. The lay shaft may either be carried on the horizontal battery sills close to the ground, or it may be carried high up on a portion of the framing; it may be either in front of or behind the battery.

Wooden Frames are very largely used, especially in American built mills, which are nearly always driven by pulleys. If circumstances admit of the employment of wooden frames, these are perhaps the most generally satisfactory. As regards the most suitable timber, almost any wood that can be obtained in beams of the large dimensions required may be used. Pitch pine answers well, but is heavy; almost any pine-wood may be used, sugar pine and yellow pine being often selected in California, whilst Oregon pine is largely exported to other countries for this purpose. Norway or Dantzig pine also answers perfectly well. Sometimes the superstructure is of light pine whilst the foundation timbers are of pitch pine or of Karri wood. When there is no construction timber at the spot where the mill has to be erected, and the entire mill has to be imported, cast-iron frames are perhaps the best, and where transport is difficult and troublesome, steel frames may be used.

When the lay shaft is low down, the so-called *A* frame is mostly employed. In this system of framing the upright is strengthened by diagonal struts and hog chains or tie bolts.

The two principal patterns are shown in Figs. 46 and 47, the chief difference being that in Fig. 46 the strut is turned to the back and in Fig. 47 to the front of the battery; of course the lay shaft must be on the same side of the upright as the strut, since one of the main functions

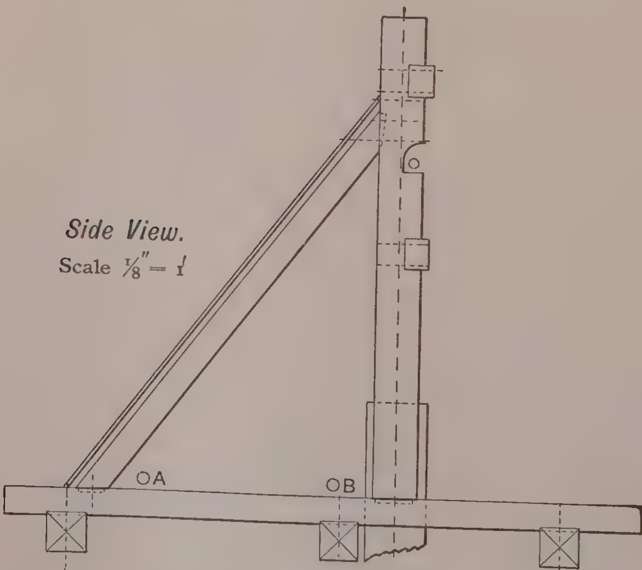


FIG. 46.

of the strut is to brace the upright against the pull of the belt.

If a mill be worked with mechanical ore-feeders, the former is probably the better arrangement, as it leaves the tables in front of the mill entirely open and free to observation from every part of the mill building. The lay shaft may be either at *A*, where it is comparatively out

of the way, or at *B*, a belt tightener being required in the latter case, as the belt will be too short to exercise sufficient pull without one. Of course the struts and belts make it difficult to get about on the feed side of the mill, but as machine feeders once adjusted require but little attention, this is not a matter of much importance.

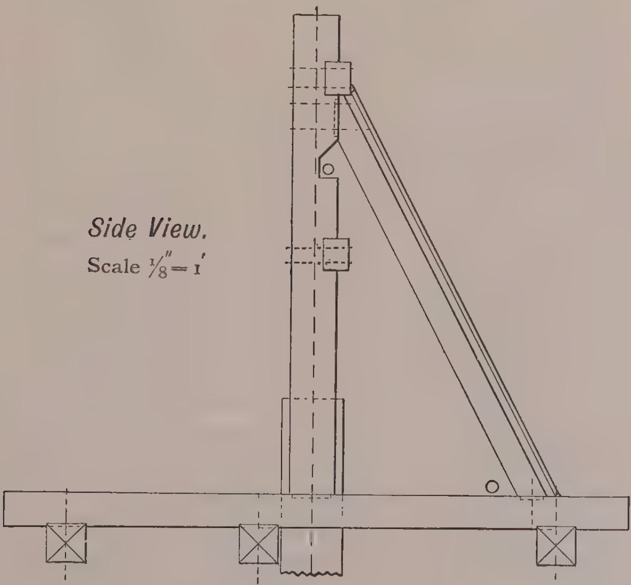
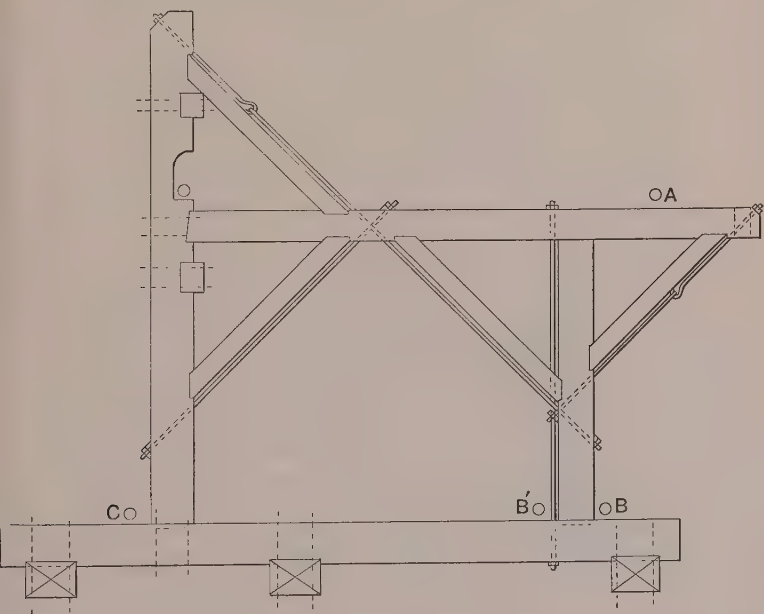


FIG. 47.

When, on the other hand, machine feeders are not used, as is still sometimes the case with small mills, it is important that the back portion of the mill should be as unencumbered as possible, in order to allow the workman who attends to the feeding to move about as rapidly and freely as possible. It will be noticed that in both

patterns the battery sills rest upon three mud sills which are buried in the ground and may be held in place by hold-down bolts. These mud sills should be (for an 800 lb. stamp-mill) about 16 inches square, or they may with advantage be 15 inches wide by 18 inches deep, running the full length of the mill. The battery sills should be about 15 inches square, and checked about 2 inches deep into the mud sills, the two baulks being held together by a couple of $1\frac{1}{4}$-inch bolts at each joint. The battery uprights or battery posts should be at least 20 inches by 15 inches. For a ten-head stamp-mill, the middle battery upright and battery sill may with advantage be a little wider than the outside ones, as it takes a greater strain. The posts should be mortised at least 3 inches into the sills, care being taken that all the joints are very carefully fitted. The mud sills should be coated with a good layer of pitch put on hot, and the rest of the frame either well tarred, or painted with three coats of good paint. The struts should be about 12 inches by 10 inches, and well mortised into both the uprights and the sills. The hog chains should be made of $1\frac{1}{2}$-inch to $1\frac{3}{4}$-inch round iron, and should be arranged for tightening up either by means of swivels in the middle or by long screw threads and large-sized nuts at the upper ends. Unless the frame is carefully and substantially constructed of strong sound timber, good results in working are impossible. The weight of an *A* frame complete for a ten-head mill of 750 lb. stamps may be taken as between 6 and 7 tons. These *A* frames do very well for small and light mills, but are not to be recommended for stamps of over 750 lbs. weight. They have been almost entirely replaced of late years by the so called knee frame, the arrangement of which is shown

in Fig. 48, which represents the general arrangement of a mill thus constructed. The mud sills and battery sills should be of at least the same dimensions as given above. The battery uprights should be about 14 inches

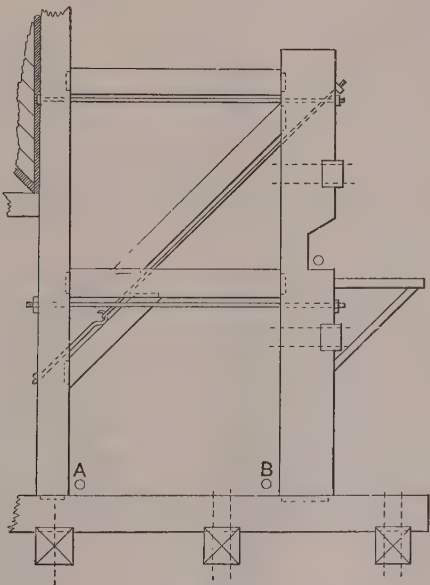


Side View
Scale 1/8" to 1 ft.
FIG. 48.

by 24 inches; the central one of a ten-head battery may with advantage, together with the battery sill on which it stands, be widened to 18 inches. The knee posts and knee beams may be about 14 inches square or else 12 by 16 inches. The angle braces should be about 10

to 12 inches square. The timbers should be mortised and bolted together as shown. A cast-iron bracket may with advantage be bolted to the battery uprights to support and stiffen the knee beam. In this arrangement the lay shaft is nearly always carried, as shown at *A*, on the knee beams, level with the cam shaft. Sometimes, however, the lay shaft is carried on the battery sill near the knee post, as shown at *B* and *B'*, or else behind the battery upright, as at *C*. Every position of the lay shaft has its attendant disadvantages and advantages. When carried on the cam shaft level, the belt has the advantage of being horizontal and well out of the way of the mill man. The lay shaft too is out of the way whilst its bearings are readily accessible. At the same time the heavy frame which carries it to some extent obstructs a clear view of the tables and also shuts out much light from them; moreover, in spite of all precautions, the lay shaft bearings are never as solid as when these bearings are directly on the battery sills. When these bearings are carried as at *C*, and therefore near the mortar box, the shaft is apt to be cut or damaged, as it is very difficult to keep it clean from fine dust, splashes of pulp, &c.; moreover, it is then usually in a very dark place, and receives accordingly less care and attention than it would if placed where it is constantly in view. The vertical pull of the belt is also an objection, and the belt is generally so short that a tightener must be used, which rapidly wears the belt, so that its life is a comparatively short one. These objections apply partially but with less force to those cases in which the lay shaft is carried on the battery sill, as in either of the positions marked *B* and *B'*, the former being the better place. The chief objection to this

latter system is that the belts are very much in the way of the mill man, and prevent ready access to the tables. The knee frame is well adapted to carrying the lay shaft when the latter transmits power to the cam shaft by means of gearing. The weight of the woodwork of a knee frame



Side View
Scale ⅛″ = 1 ft.
FIG. 49.

complete for a twenty-stamp mill is about 16 to 17 tons. Another form of frame which has come greatly into favour is the reversed knee frame, shown in Fig. 49. In this system the battery posts are braced against the front uprights of the ore-bin frames, strengthened at times by

diagonal braces, so as to make a very strong, compact

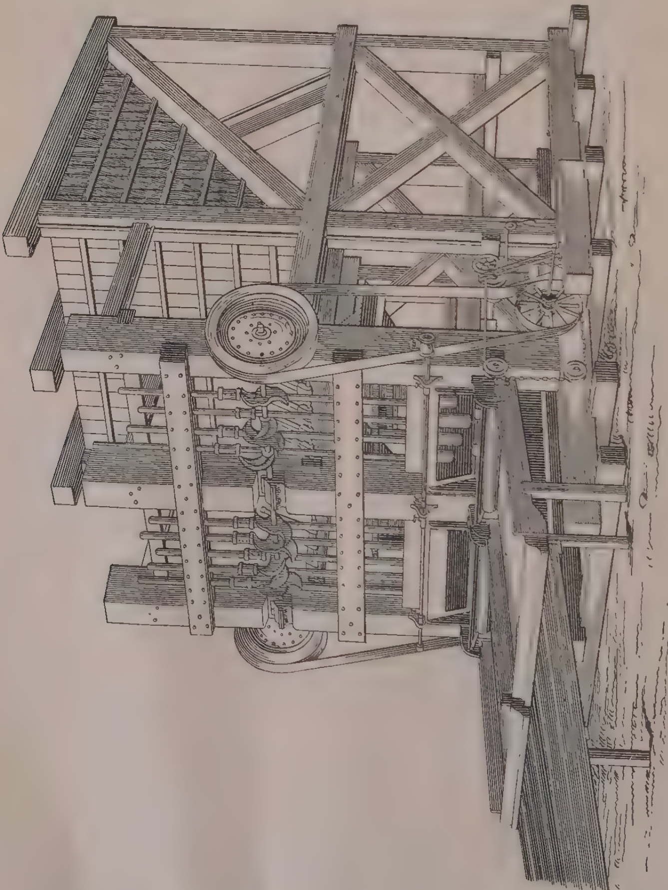
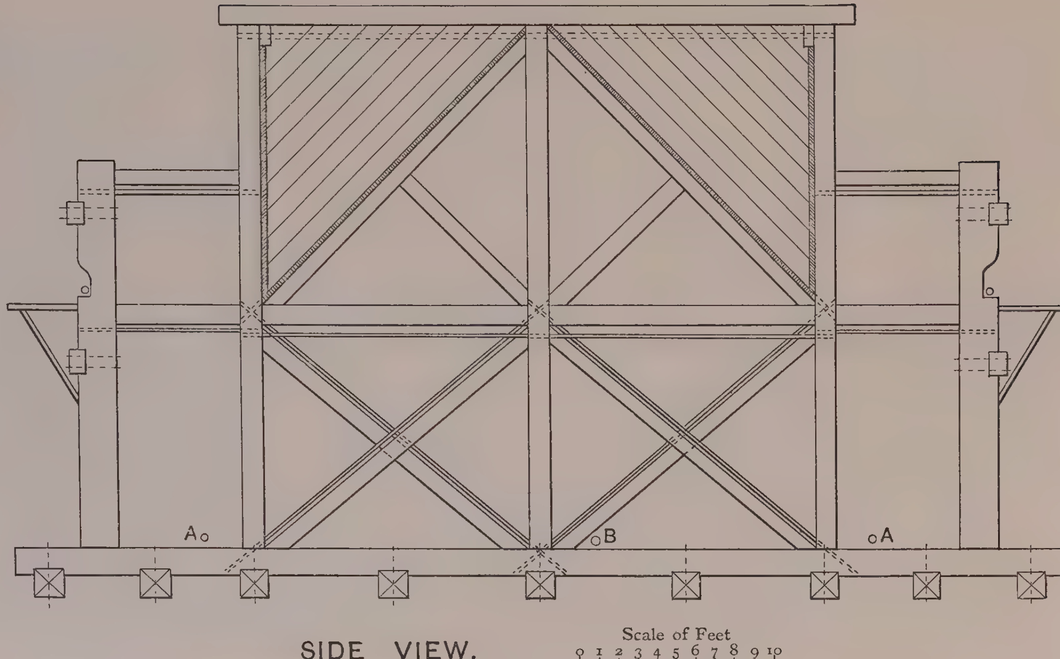


FIG. 50.

frame. The only positions available for the lay shaft are at *A* and *B*, and it is the sole valid objection to this



SIDE VIEW.

FIG. 51.

frame that it must carry this shaft in a somewhat inconvenient place, and as the distance of the battery posts from the ore-bin uprights is restricted by the arrangement of the ore-bins and ore-feeders, the belt is necessarily rather short; nevertheless, when the shaft is set close to the ore-bin uprights as at *A*, the belt is frequently just long enough not to need any tightener, although one is mostly employed, as shown in Fig. 50. This style of frame is also very well adapted to driving by gearing, the lay shaft being in that case behind the battery. The great advantage of this frame is that it affords an uninterrupted view over the entire table floor of the mill, and presents no obstacle to the moving about of the mill men on that side; it can scarcely, however, be used to advantage unless mechanical ore-feeders are employed. A perspective view of a ten-head battery arranged on this system with reversed knee frame without diagonal braces, having its lay shaft close to the battery uprights, and using a neat and strong belt tightener, is shown in Fig. 50, the mill in question having been built by the Sandycroft Foundry Company, Limited.

The reversed knee frame is also the most suitable for the design of back-to-back mills, an arrangement that is becoming popular for many of the large mills that are rendered necessary by the economic conditions of modern gold mining, though it should never be applied to a smaller number of stamps than 80. The framing of such a back-to-back mill, together with that of its ore-bins is shown in Fig. 51, this mill being intended for operation in conjunction with an independent rock-breaker house, a couple of tracks from which should be carried over the top of the bins. The scantlings of the timbers should be about the same as given for knee frames; it must not

be forgotten that the ore-bin sills have to carry the whole weight of the ore in the bins, and should therefore be of ample strength, and their mud sills securely bedded. There may be two separate lay shafts as at *A*, but it is perhaps better to drive the entire mill off one main line of shafting at *B*.

Steel Frames.—Wrought-iron frames have occasionally been used for mills, but this material has practically been replaced by steel, which is in every way more suitable for the purpose. Even steel, however, forms by no means a satisfactory frame, as it is much too elastic and admits of too much vibration. Its special advantage is that of portability. By replacing wooden beams by compound girders to be riveted or bolted together on the spot of erection, the framing of a heavy mill may be carried in comparatively small pieces. The mud sills are sometimes made of wood, but more often are replaced by blocks of concrete or of brick or stone laid in cement, to which the battery sills and the rest of the frame are bolted by strong hold-down bolts. A usual type of steel frame is shown in Fig. 52, which represents a twenty-head battery with iron frames, shown in perspective. It will be noted that in this no attempt has been made to dispose the metal to special advantage, it being a simple *A* frame in which the wooden beams have been replaced by compound girders consisting each of two lengths of channel iron held together by bolts with suitable distance pieces. A rather better design is shown in Fig. 53, where the frame is constructed of similar compound girders, but so arranged as to give greater stiffness. Both these mills were designed and constructed by Messrs. Bowes Scott and Western, Ltd., of London. Steel frames are, however, whenever pos-

sible, to be avoided. The weight of a steel frame for a ten-head mill may be taken at about five tons. In con-

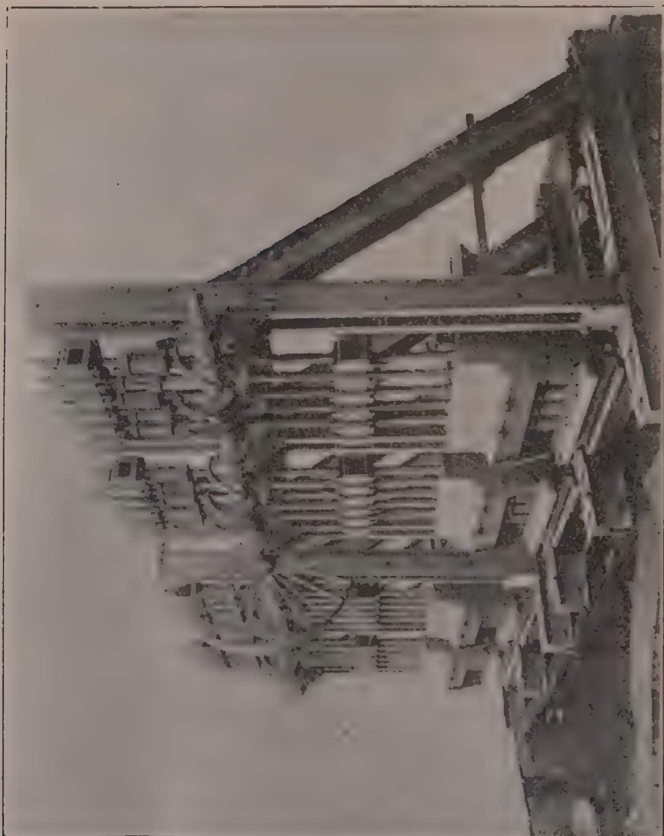


FIG. 52.

paring these figures with the weights of wooden frames, it must not be forgotten that the former exclude, whilst the latter include mud sills, which form nearly a quarter

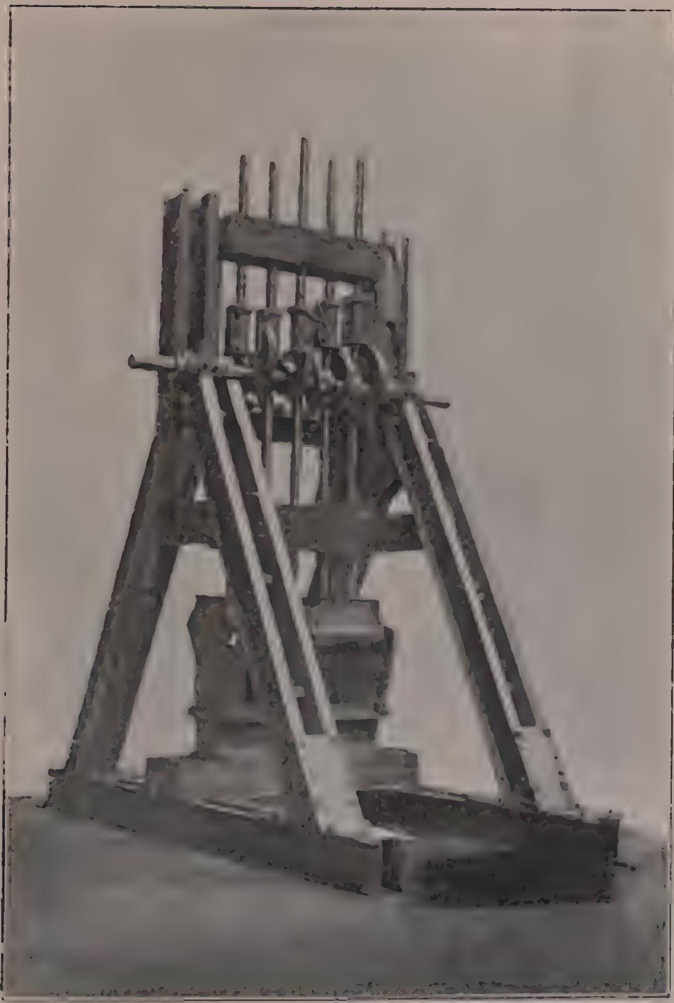


Fig 53

of the total weight of the wooden frames. I believe that the first steel frames made were designed in 1879 for a small composite portable mill built by Messrs. Appleby Bros. for Effuenta, Gold Coast of Africa, in which no part exceeded 150 lbs. in weight.

As far as I know, cast-steel frames have never yet been employed; in the present advanced condition of the art of steel casting, and with the comparatively low prices for which steel castings can now be bought, their construction should offer no difficulty, and it would seem difficult to find any better material, whenever lightness is an especial consideration.

Cast-Iron Frames have been largely used in Australia, and to some extent also on the continent of Europe. In America and in this country they have never been much used, although there is a very great deal to be said in favour of this material. It is sufficiently stiff to form a good frame, and elastic enough to stand the jar of the stamps. Colonial-built mills with cast-iron frames have been running in some places for over 30 years, and are in as good shape, whilst they have cost quite as little for repairs, as the best wooden frames. The castings need not be of unwieldy size, and can be made in pieces capable of being bolted together at their destination without greatly weakening the frame. Moreover, a properly designed cast-iron frame is fairly self-contained, and can dispense with the angle struts used in wooden and steel frames. The best system is probably some form of hollow casting, copying more or less the frame of a steam hammer. The cast-iron frames for a ten-stamp mill of 800 lb. stamps weigh about three tons complete (of course without mud sills or foundation blocks), so that this is by no means a heavy system of framing. A

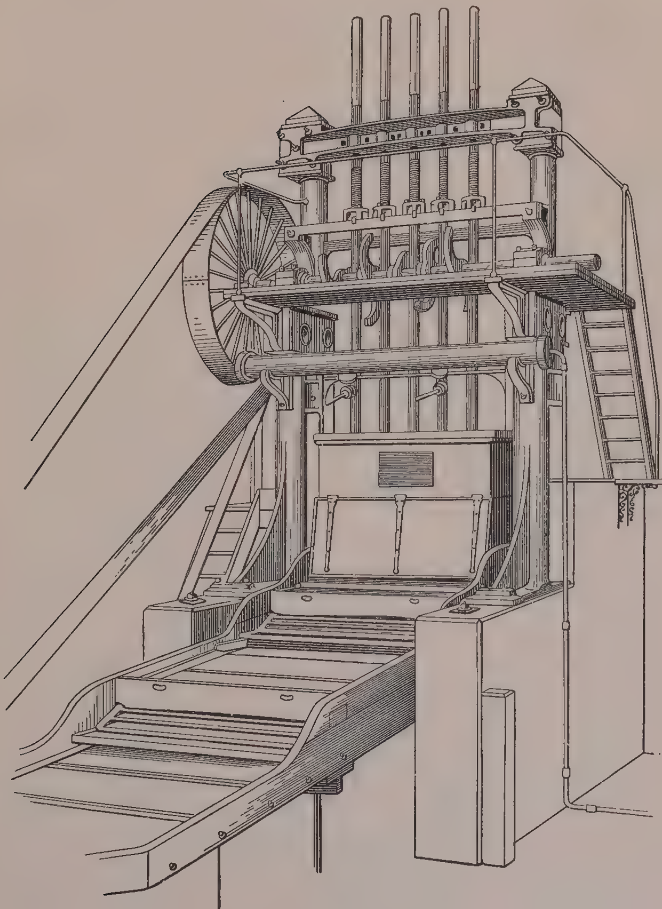


FIG. 54.

very neat pattern of cast-iron frame is shown in Fig. 54, which gives a view of a five stamp mill built in Queensland; it was erected at the Colonial and Indian Exhibition of 1886 in London, and may be taken as representing the best colonial design. The figure is reproduced from a photograph kindly supplied by the Queensland Government Office. The details of the framing are shown in Fig. 55.

A somewhat similar pattern having two cast-iron columns in the lower part, making a very good, stiff frame, is built by Messrs. Thompson and Co., of Castlemaine, Victoria.

A new cast-iron frame, recently designed by Messrs. Bowes Scott and Western, Ltd., stiffened, however, by a steel strut that ought hardly to be required, is shown in Fig. 56.

Lay Shafts.—The proper positions for these have already been discussed under the various types of frames; with steel and iron frames they are best carried on massive bearings bolted down to heavy blocks of masonry or concrete. As the lay shaft runs usually at from twice to thrice the speed of the cam shaft, and has no strains to bear except those set up by the transmission of power, it can be correspondingly lighter. In a long mill, however, the end of the shaft that is nearest the prime mover has to transmit the power for driving all the cam shafts, and must be strong in proportion. As power is taken off for each successive mill pulley, the diameter of each successive section of the lay shaft can be correspondingly reduced; much weight is thus saved, although the cost of patterns and machining may be a trifle increased, whilst a bigger stock of spare parts such as brasses, couplings, &c., must also be carried. The

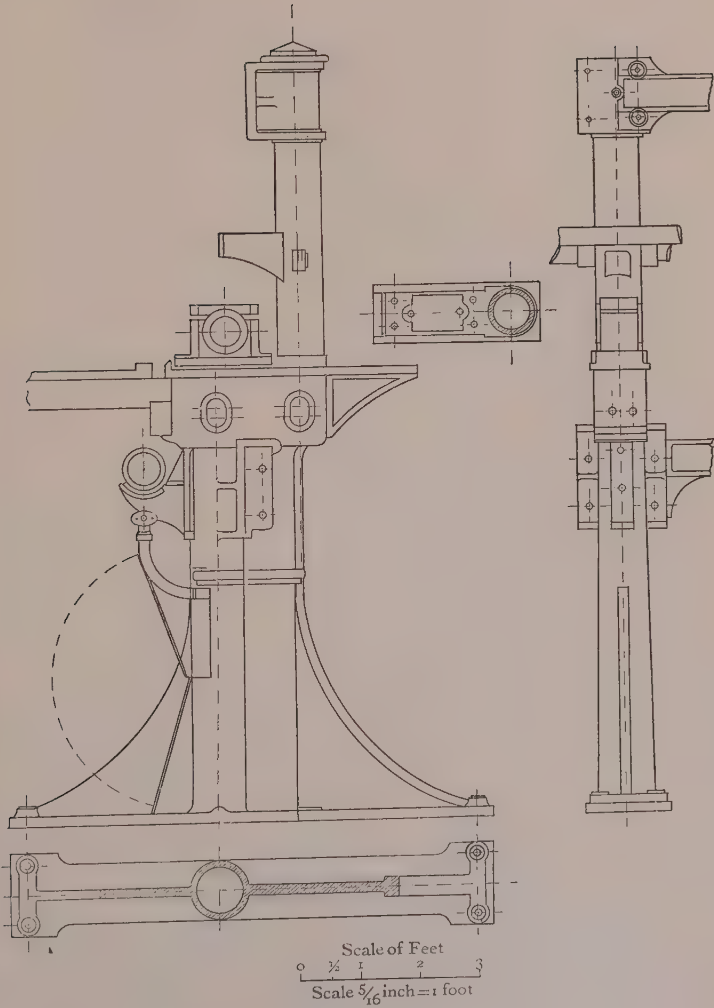
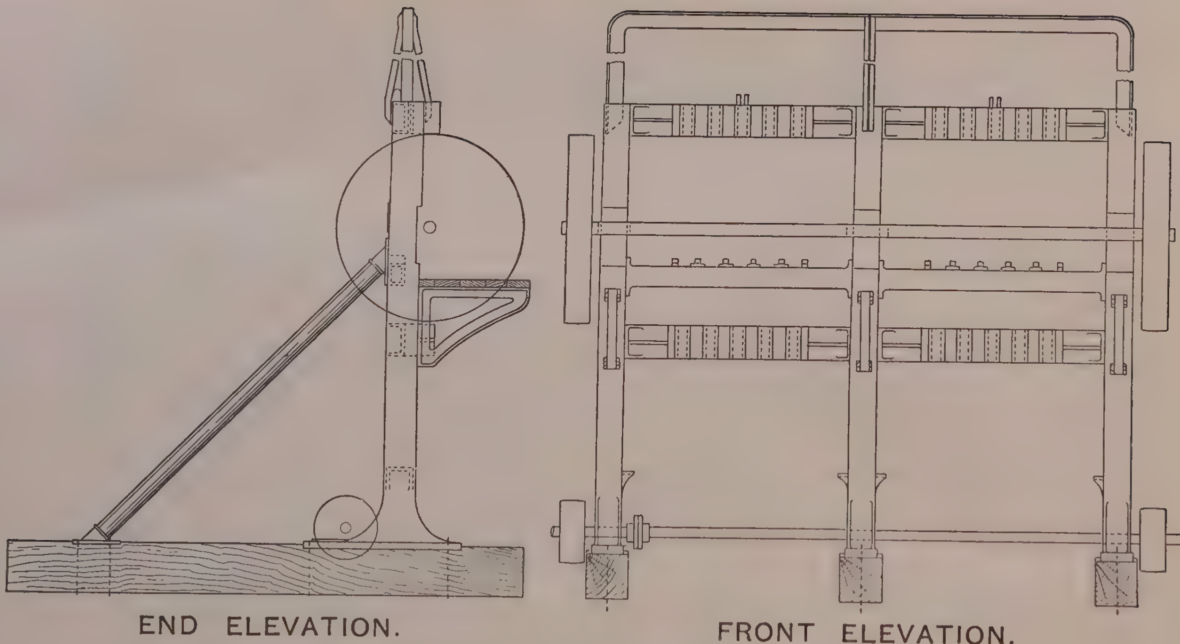


FIG. 55.



Scale, $\frac{1}{6}$ inch = 1 foot.

Fig. 56.

lay shaft is occasionally coupled direct to the prime mover, but is usually driven by belting or more rarely by gearing; for a mill of any size cotton driving ropes are undoubtedly the best.

In small or medium-sized mills the motive power is best placed at one end of the lay shaft, and is often so situated even in large mills, although this position tends to set up severe lateral strains in a long line of shafting; in such cases the driving power should be applied in the centre of the lay shaft. The driving pulleys on the lay shaft should be of the ordinary type; they should be provided with either ordinary clutches or friction clutches,

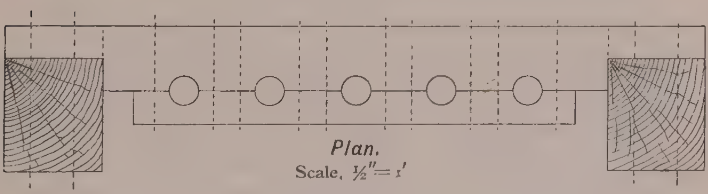


FIG. 57.

unless there is a tightener on the battery belt, so as to enable any one battery to be stopped or started independently of the others.

Guides.—The frames of a battery are always connected by a pair of horizontal guide-beams, which are bolted to them, and which serve not only to stiffen the frame but also to carry the guides within which the stamp stems work up and down. Their function is to withstand the tendency of the stamp stem to deviate from the vertical, as already explained. In their simplest form the guides, form a part of the guide-beam. In the case of wooden

frames, this arrangement is shown in Fig. 57, this being a plan of one complete guide-beam, the battery posts at either end being shown in section.

It will be seen that one half of each stem works in the guide-beam and the other half in a single cap piece, which is bolted to the beam by a number of 1-inch bolts (indicated by dotted lines), its exact position being adjusted by means of these bolts and by hard-wood distance pieces or wedges (not shown in Fig. 57), driven down between the two beams to keep the stamp stem from being jammed. This arrangement is unsatisfactory, because, first of all, the motion of the stamp stem takes place across the grain of the wood, causing unnecessary friction and wearing the guides out rapidly. Moreover, it is impossible to adjust any individual stem that may require it; if, for the sake of economy, it is thought desirable to adopt this plain guide, there should be a square recess cut out round each stem and a bushing of hard wood driven down for the stem to work in. This bushing can be taken out and renewed when worn, and the grain of it will be vertical so as to be parallel to the motion of the stem. Hancock's guides, made by the Sandycroft Foundry Company, Ltd., are quite similar, except that the hard-wood bushing has a projecting square shoulder, to keep it from shifting up or down. With iron frames a somewhat similar arrangement in cast-iron is also at times adopted, only in this case, as shown in Fig. 58, each stem has usually its own separate cap. Mostly cast-iron guides are simply bored to fit the stems, but they are preferably lined with Babbitt metal. Metal guides are not, however, to be recommended under any circumstances; they are apt to work hot and to cut the stems, and their lubrication is less satisfactory than

that of wooden guides. Moreover, with wooden guides practically all the wear is confined to the guide, which is easily and cheaply replaceable, instead of its being shared by the more costly stamp stems.

Messrs. Fraser and Chalmers, Ltd., make two patented patterns of stamp guides, the Fargo and the Broughall, either of which is quite satisfactory, though specially adapted to wood-framed mills only. In these guides each stem has its independent pair of wooden bushes which are tightened up by wedge-shaped iron keys and nuts in the former, and by clamping screws and strapping plates in the latter, which is here illustrated in Fig. 59. Both these guides have the advantage that each stamp can be adjusted, and its guide-blocks loosened or tightened without interfering with the others, so that perfect verticality of the stamp stems can be maintained and wear of the bushes taken up without stopping the running of the mill; the bushings of any one stamp

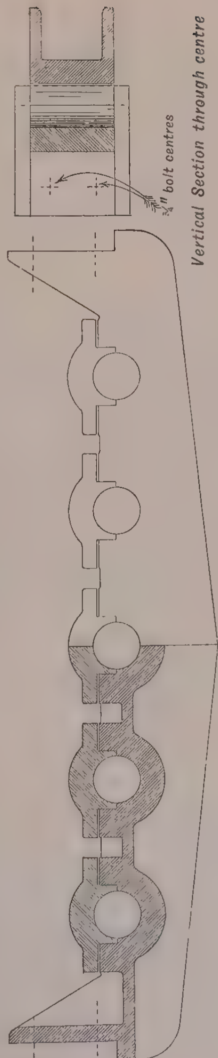


FIG. 58.

can also be taken out entirely, and the stamp drawn up for repairs or replacement without touching any of the others. The objections to them are their expense and their complication, each consisting of a number of small pieces. The most satisfactory pattern of guide yet designed is the one shown in Fig. 60, which sufficiently explains its construction. This is in very general use on the Pacific Coast and can be thoroughly recommended; it can be used

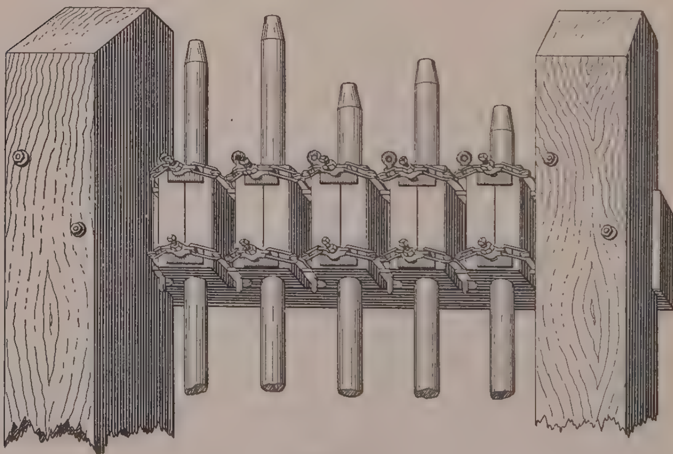


FIG. 59.

with either wooden or iron guide-beams, and has all the advantages above enumerated besides that of having very few loose parts, which is a matter of great importance in a stamp-mill. All these three last-named systems of guides have the advantage that the grain of the guide-bushing is parallel to the direction of motion of the stamp stem. Guides are usually from 12 inches to 16 inches deep. Very often the lower guide is made

a little deeper than the upper one, but there is no mechanical justification for this practice, as the strains on each are practically the same. Especially where any of the systems of loose guide-blocks here recommended are in use, it is advisable that the depths of the upper and lower guides should be the same, so that the bushings shall be interchangeable, this system necessitating the maintenance of a smaller stock of spare parts. The

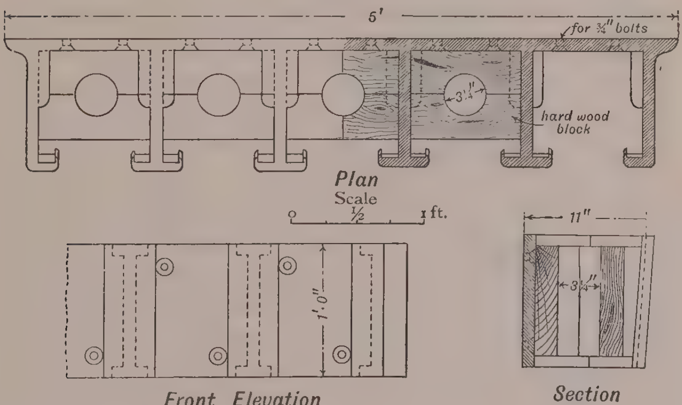


FIG. 60.

bottom of the lower guide should be about 4 inches to 6 inches above the top of the mortar-box; in the old style of wooden mortar-box the lower guides formed the cover of the mortar, but this plan has now been discontinued. There should be space enough below the lower guide to allow the mill man to get at the cover of the mortar readily. It is also well to hang a piece of old rubber belting, canvas, or oilcloth, having holes for the stamp stems to pass through, below the lower guides, to prevent

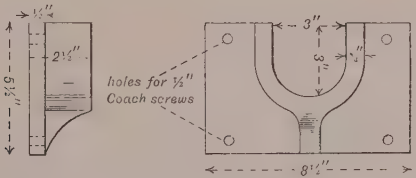
any of the lubricant used from finding its way into the mortar. From the top of the lower guide-beam to the bottom of the upper the minimum space must be the length of the cam from toe to toe plus the height of the tappet and an inch or two for clearance. This distance can therefore never be less than about 4 feet and is usually about 6 feet.

Lubrication.—It is necessary to keep the guides lubricated so as to keep the friction of the stems against them as low as possible. When iron guides are used they need practically no lubrication if they are babbitted with a good soft metal; if not, anti-friction grease may be used, but very sparingly. Soft soap is preferable, as being less injurious to the process of amalgamation, when any of the lubricant finds its way into the battery box. In the case of wooden guides the best lubricant is finely ground graphite made into a stiff paste with a little soft soap. Various lubricating compositions are sold by makers of milling machinery, but the above mixture may be thoroughly recommended as the result of extended experience.

Working Platform.—In order to enable the mill man to attend to the tappets, upper guides, &c., a working platform is usually arranged about the level of the cam shaft or a little below it. This platform may be either at the back or the front of the mill; it must run along the full length of it and must be provided with steps or ladders at either end or at any convenient point near the middle of the mill. American engineers seem to prefer to put this platform behind the mill, whilst Australians generally have it in front; it is practically unimportant on which side it is placed, except that, if in front and very broad, it is liable to cut off too much light from the

amalgamating tables below. As a good deal of heavy work has to be done on the platform, it should be carried on strong brackets, and itself made of stout plank not less than 2 inches thick; the planks ought to be well jointed so as to prevent small tools, nails, &c., from dropping through. There should be a stout railing along the outer edge of the platform.

Stamp Supports.—Occasionally it becomes necessary to "hang up" one or more of the stamps—that is to say, to support them so that the lower face of the tappet shall be about half an inch above the travel of the cam,



Side View. Scale, 1½" = 1′ *Front View.*

Fig. 61.

in order that the cam shaft can continue to revolve without touching the stamp or stamps hung up. The American method of doing this is by means of a jack-shaft and finger bars. The jack-shaft consists of a piece of shafting about 3 inches in diameter, of exactly the same length as the space between the battery uprights. On each of the inner faces of the battery uprights is screwed a bracket such as shown in Fig. 61; this is best countersunk into the face of the timber and secured by four 8-inch coach screws. These brackets carry the jack-shaft, which need not be turned bright. A piece of cold-rolled shafting answers capitally; it must be strong

enough to carry the weight of all the five stamps without permanent deformation. The fingers, a usual pattern of which is shown in Fig. 62, consist of a saddle-shaped

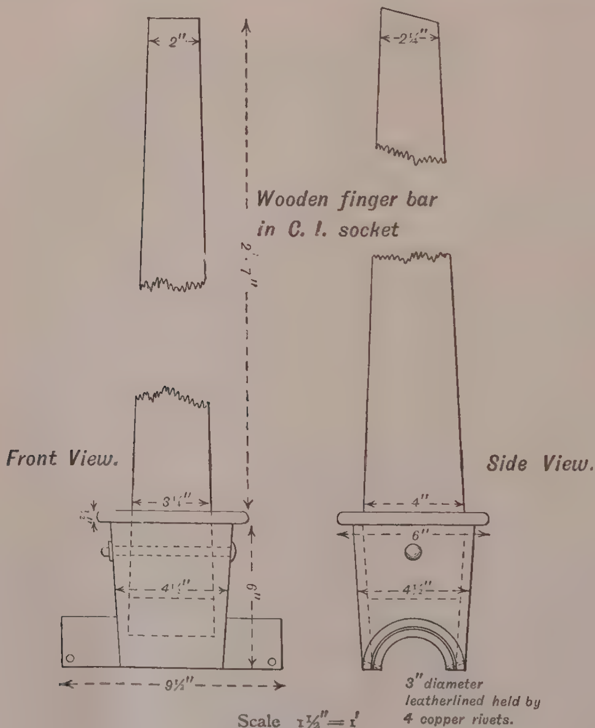


FIG. 62.

casting which is lined with leather and rests on the jack-shaft, terminating in a socket about 3 inches square into which fits the wooden finger, of requisite length to support the tappet clear of the travel of the cam. These

fingers are usually strapped with $\frac{1}{8}$-inch sheet-iron at the point, and have an iron handle. When not in use these fingers lean back against the edge of the working platform, which is often notched to receive them. When it becomes necessary to hang up a stamp, the mill man takes in his hand a fid stick, which is best made of a number of strips of leather or rubber belting 3 inches wide, 2 inches deep, and 18 inches long, furnished with a wooden handle at one end (or the whole stick may be made of wood if desired). He lays this fid stick on the cam as it revolves, keeping a firm hold of the handle,

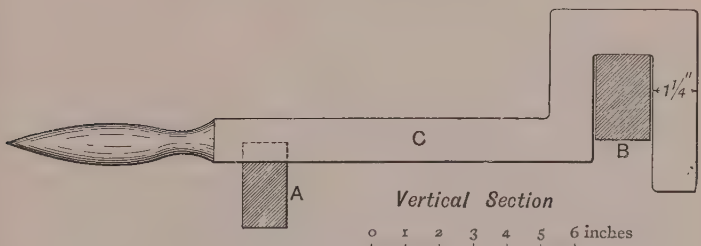


FIG. 63.

and then presses his knee firmly against the finger bar. The cam lifts the tappet to a greater height than usual by the extra thickness of the fid stick, and is thus lifted so high that the tappet just clears the finger bar, which is pushed underneath it by the mill man, who then pulls back the fid stick. The operation is a simple one, but requires care and practice; it is facilitated if the platform is so placed that the cam is revolving towards the operator. If the stamp has to remain hung up for any length of time, it is as well to put a lashing through the handle of the finger bar round both the bar and the

stamp stem, so as to obviate any risk of the former slipping aside or of the latter being jarred off. In Australia the above arrangement is replaced by two iron bars running lengthways along the battery, and a short finger bar that rests across these at such a height as just to support the tappet. These bars, A and B, Fig. 63, are about $2\frac{1}{2}$ to $3\frac{1}{2}$ inches deep and $1\frac{1}{4}$ to $1\frac{1}{2}$ inches broad, and are firmly bolted to the battery uprights; the back bar, B, is 2 to 3 inches higher than the front one. The finger bar, C, here made of iron about $1\frac{1}{4}$ inch square, is bent so as to catch on the back bar, and when in position just supports the tappet at the desired height. The front bar is notched, so that once the finger has been slipped into its place it cannot be jerked away again by the jar of the stamps. If there are no notches, the finger bar should be lashed to the stamp stem. It is scarcely necessary to say that this bar is on the side of the stem remote from the cam. The finger bars are usually kept out of the way close to the uprights, or else are hung up in some convenient spot close at hand until they are required. This makes a very good and compact arrangement for hanging up stamps, is quite as effective as the American method, and takes up less room.

Stamp Hoists.—It occasionally becomes necessary to hoist a stamp up altogether, as, for instance, when a tappet slips or a stem breaks. This is almost always done by a chain-block, which is far the most convenient plan; a 15 cwt. chain-block does perfectly well. It is usually carried by a little traveller or crawl, which runs on a rail above the stamps. A crawl which answers this purpose thoroughly is shown in Fig. 64. It will be noticed that this pattern requires two rails. Some are

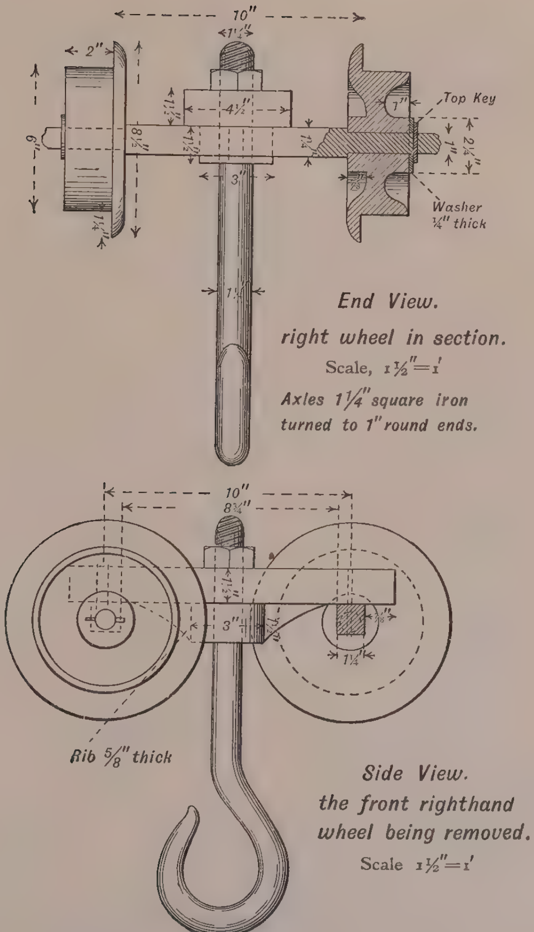

10"
2"
Top Key
Washer
¼" thick
End View.
right wheel in section.
Scale, 1½"=1'
Axles 1¼" square iron
turned to 1" round ends.
10"
8¾"
Rib ⅝" thick
Side View.
the front righthand
wheel being removed.
Scale 1½"=1'

FIG. 64.

made requiring only one. The rails may be suspended from the framing of the mill-building, or they may be supported on the battery uprights, as shown in detail in Figs. 65 and 66, the latter system being the more

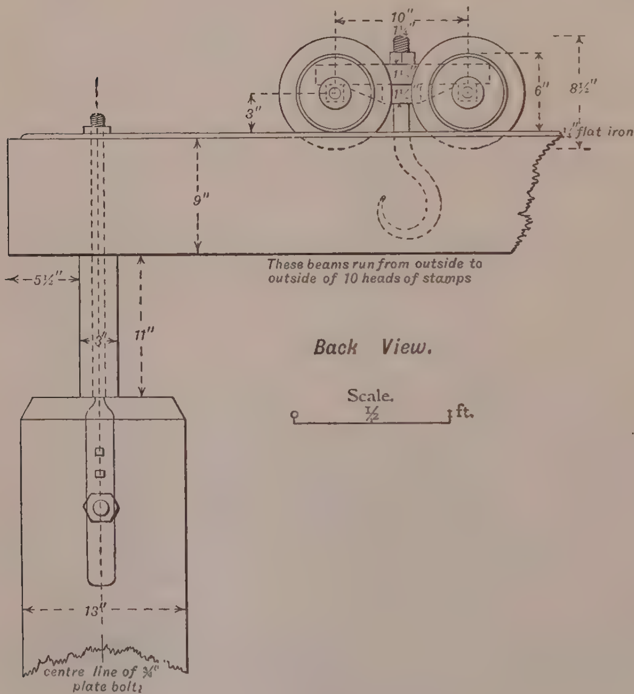


Fig. 65.

reliable, although it is liable to the objection of allowing too little head room. In order to attach the hook of the chain-block to the stamp, some makers drill a hole into the ends of the stem and tap it, so that an eye

bolt can be screwed into it when required. This is not a good arrangement, as the hole is apt to get filled with dirt or the thread to be burred up. A better plan is to slip a ring such as shown in Fig. 67 over the stem; as

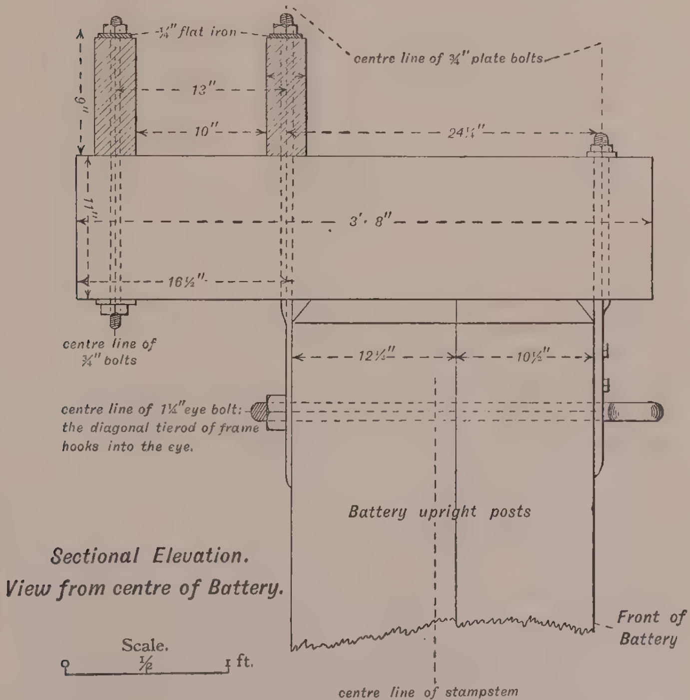


FIG. 66.

the pull of the tackle is somewhat to one side, this ring, which is merely a roughly made forging, grips the stem quite firmly and securely by friction alone; sometimes an additional friction grip is attached to the ring, but it

is unnecessary. The ring alone answers all purposes perfectly.

Water Supply.—There should be a liberal water supply of clean water to the mill whenever it is possible to obtain this desideratum. The principal water-main should run the entire length of the mill in any convenient position, branches being carried to each separate battery. It is by no means a matter of indifference how the water

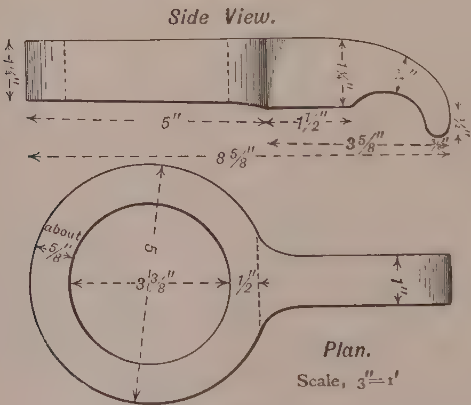


FIG. 67.

is led into the mortar box. There should be two pipes to each box, usually from $\frac{1}{2}$ inch to 1 inch in diameter; they should discharge opposite the centres of the second and fourth stamps, and the streams should be directed against the back side of the box. Each pipe must have its own separate valve, which can be regulated from the front of the battery, where the requirements of the mill can be at once ascertained. The water supply should be so adjusted as to keep both ends of the mortar uni-

formly supplied, and to prevent the crushed quartz from packing at one end or the other. Sufficient water must be used to carry all the quartz out of the mortar box as soon as it is crushed, and keep it suspended as a fairly thick pulp inside the mortar. The level of the water in the box should stand at from $2\frac{1}{2}$ to 5 inches above the level of discharge of the screen. If the pulp requires to be diluted for the operations outside the battery, the necessary water must be added outside. A length of perforated pipe running along just outside the screens is often used for this purpose. In such a case, this perforated pipe should form a movable arm so that it can be turned out of the way when the screen frame has to be taken out. For each battery there should also be a separate branch pipe fitted with a valve and carrying a length of hose with an iron nozzle for washing down the tables, &c. There should be other branches for washing down the floor of the mill and for miscellaneous purposes generally. When there is a plentiful water supply, it is always good practice to have branches laid on to all parts of the mill-building. The amount of water consumed by a battery varies within very wide limits according to the nature of the ore, the degree of fineness to which it has to be crushed, the mode of treatment adopted for it, &c.; it may be taken as between 80 and 320 cubic feet of water per ton of ore crushed, averaging, say, 150 to 200 cubic feet per ton of ore. Reckoning each stamp to crush 3 to 4 tons per twenty-four hours, the supply of water required for a five-stamp battery may be taken at $1\frac{1}{2}$ to 2 cubic feet of water per minute on the average, or say about 20 cubic feet per stamp-head per hour. Battery pulp will thus consist of about 10 per cent. by weight of quartz

and 90 per cent. of water. When the natural water supply is scanty, it will be necessary to run the tailings from the mill into large settling pits, where the particles of sand are allowed to settle, and the clean water is then pumped back again to the tanks that supply the mill. By this means fully half of the total requirements of the mill may be supplied. If the water is very muddy, it should either be allowed to settle as completely as possible in large pits fitted with numerous baffle plates, or else filtered through a brushwood filter or some similar contrivance. It is of the utmost importance that the mill should be supplied with the cleanest water possible. If acid, as may happen with certain mine waters, or with water that has been in contact with decomposing pyritic matter, lime or wood-ashes may be thrown into the supply tank with much advantage. In a few instances sea-water has been used in the battery, without injurious effect; brackish water is similarly used in West Australia.

Flooring.—A stamp-mill should always be floored very substantially, and with some material that will not allow of leakage either of water or of quicksilver. The best plan is to put in a good cement floor, or, failing this, a floor of stout 3-inch plank having the joints carefully caulked like the deck of a ship. The floor ought to have a gentle uniform slope downwards from the ore-bin end. At the lower end of the floor there should be a gutter delivering into a tank sunk in the floor. The floor should be washed down with a good stream of water at least once a day, and all the dirt, &c., thus collected in the tank put aside for further treatment, as it is sure to contain gold and probably quicksilver also.

Bins.—Every mill ought to be provided with bin capacity sufficient to hold at least a twenty-four hours' supply of quartz, and more if possible. These bins are usually triangular in vertical section, terminating at the lower end by shoots leading one to each mortar box. Where mechanical ore-feeders are used, the shoot delivers directly into the hopper of the ore-feeder. In other cases it delivers on to a platform at the back of the stamps, this platform being preferably covered with sheet-iron. The aperture of the bin ought to be closed by a gate regulated by a rack and pinion or a screw, and fitted with a proper hand wheel so that the gate can be set at any given depth of aperture and the delivery of quartz thus controlled by the mill man and kept uniform. Both the bin and shoot ought to be lined with heavy sheet-iron about $\frac{1}{4}$ inch thick for the former and $\frac{3}{8}$ inch for the latter, held down to the planking by flat-headed nails or screws. In large mills it is not advisable to have one bin to supply more than twenty heads of stamps. When a separate rock-breaker house is used, the tram-line that delivers the crushed ore to the mill should come in on top of the mill bins. When the rock-breakers form a part of the mill structure, the rock-breaker floor is directly above the bins, and the crushed ore from the rock-breaker as well as the fine ore that passes through the grizzly, drops direct into the battery bin. An arrangement of this kind is shown in section in Fig. 68, which illustrates the combination of this arrangement with reversed knee frames, this being a section of the sixty-stamp mill built for the Montana Company, Limited, by Messrs. Fraser and Chalmers, Limited, of Chicago. Fig. 69 shows the arrangement of rock-breaker floor and ore-bins for an ordinary knee-

framed battery. It will be noticed that in these cases there is a rock-breaker- as well as a battery-bin, so that

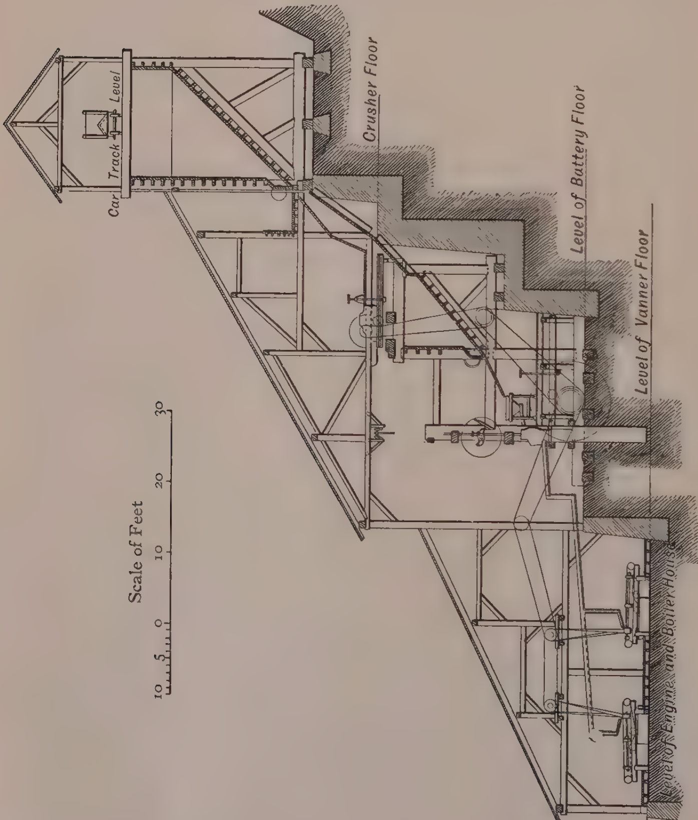


FIG. 68.

very large reserves of ore can be carried in the mill building. The battery bins of the Montana Mill hold 500 tons of ore and the rock-breaker bins 2,500, making

a total ore supply capable of keeping the mill running at full speed for over a fortnight. As a rough general rule, it may be taken that the bin capacity of a mill should

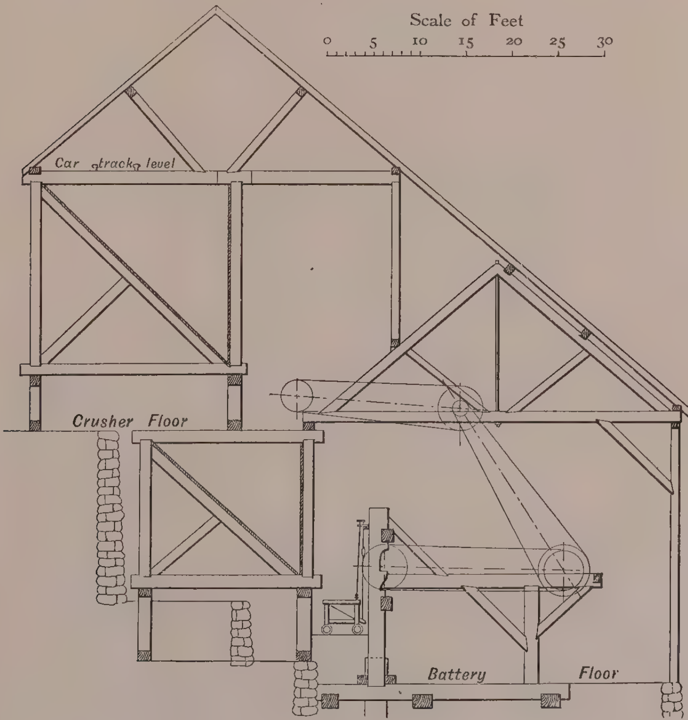
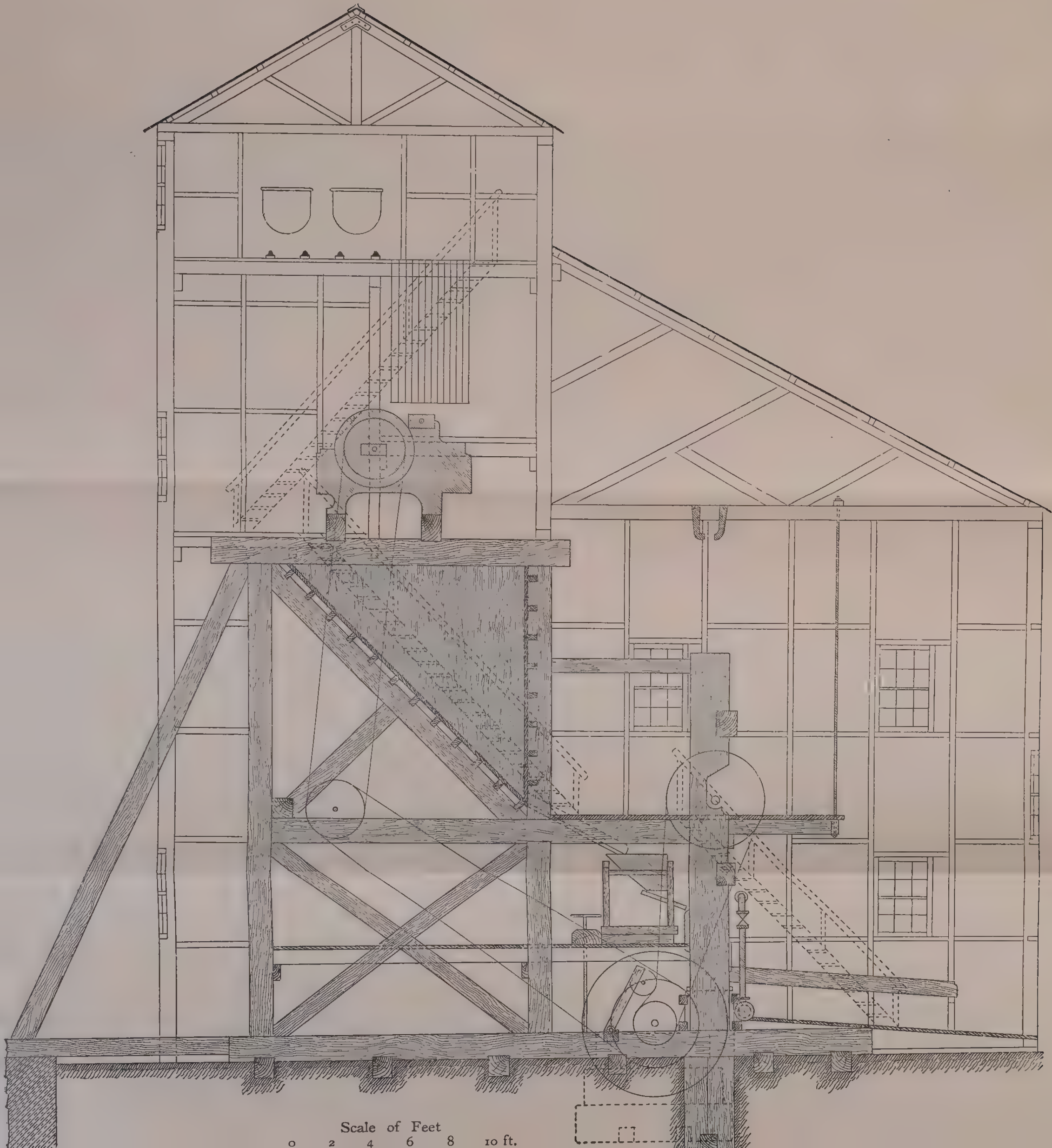


FIG. 69.

never be less than about 60 cubic feet per stamp; a bin measuring 10 feet long and 10 feet high by 6 feet deep, and having, therefore, a capacity of 300 cubic feet, will thus suffice for a five-stamp battery; but it must be

borne in mind that this is the minimum that should be allowed, and that, moreover, it is never possible to fill an ore-bin quite full up to the top. In the Witwatersrand district the rock-breaker bins are small, rarely exceeding 300 tons capacity, but the mill ore-bins are very large, having a capacity of 100 to 250 cubic feet or about 5 to 12 tons per head: the bin of the 160-stamp mill of the Glencairn Main Reef Gold Mining Company, Limited, has a total capacity of nearly 49,000 cubic feet, equal to 300 cubic feet or about 14 tons per stamp. Figs. 70 to 72 show the details of a twenty-stamp "high" mill (as the combination of mill and bin with rock-breaker above them is called) as constructed by the Sandycroft Foundry Company, Limited, and may be taken as the type of a good mill of this description.

General Arrangement.—A mill of less than sixty stamps is always arranged in one straight line, this being the most generally suitable plan. At one end of the mill-building is usually a small fitting-shop for the light repairs that are needed in the mill, and sometimes a machine shop as well. There should also be a clean-up and amalgam room and a retorting and melting room; sometimes the retorting and melting room forms part of the assay office, which must be far enough from the mill for the balances not to be affected by the jarring of the stamps. There should be a large store-room, where ample supplies of mill stores and duplicates and spare parts are kept ready for use. Large mills are usually built in groups of twenty heads, one grizzly and rock-breaker being able easily to distribute ore to four batteries, when the rock-breakers and mills are in the same building. With a separate rock-breaker house the question of ore distribution is, of course, much simplified.


Scale of Feet
0 2 4 6 8 10 ft.

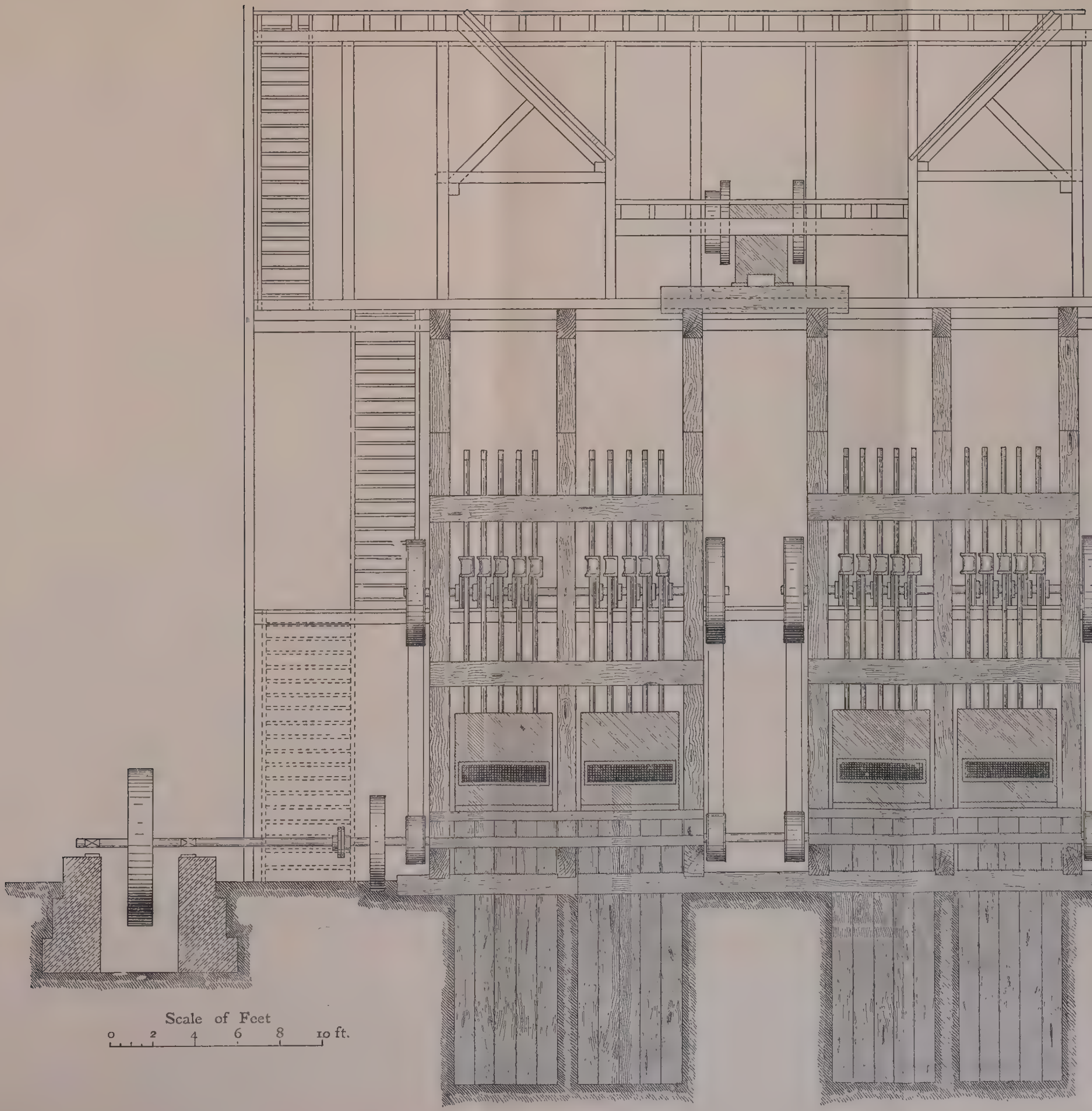
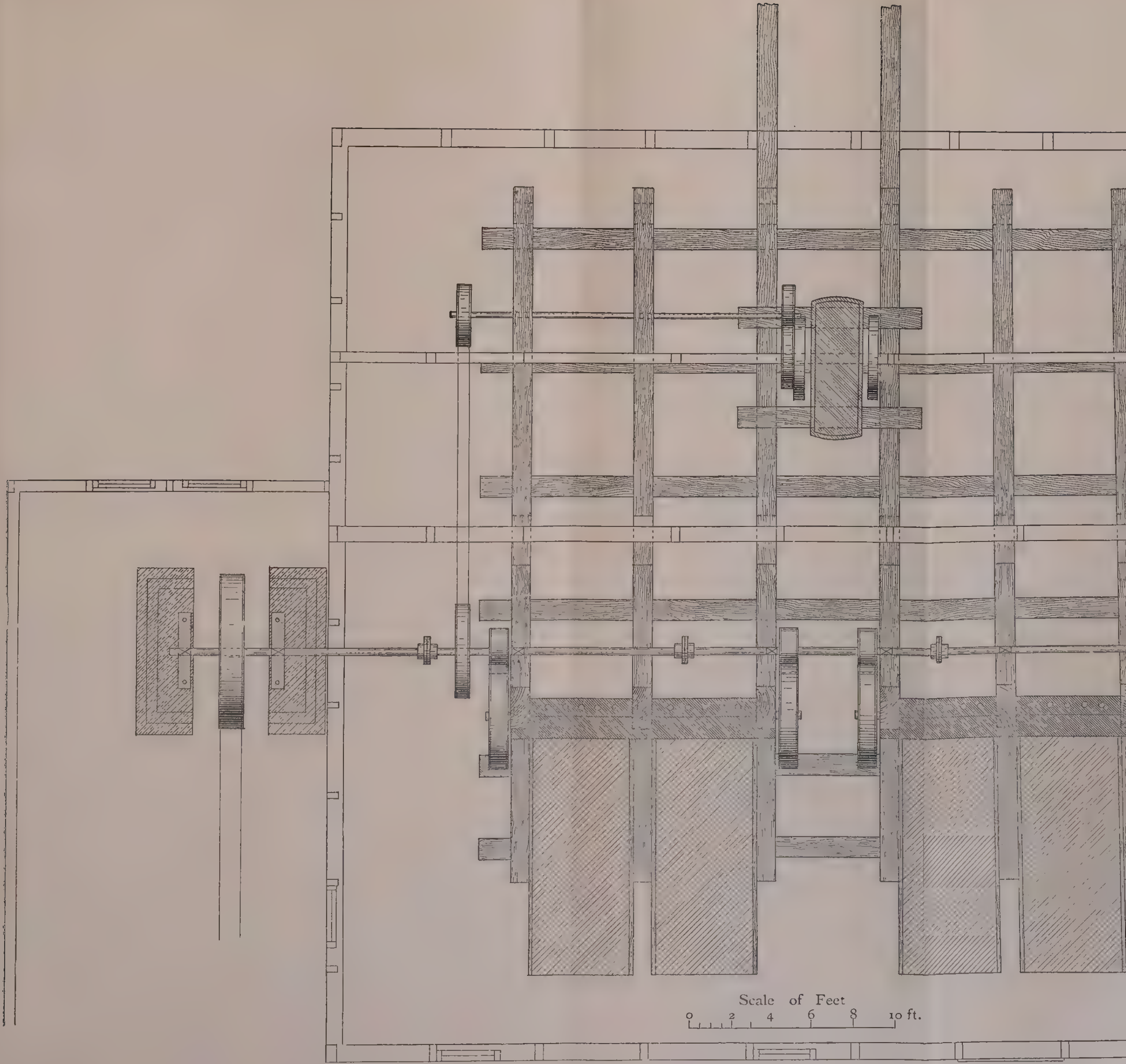


Fig. 71.



To follow Fig. 71.

Fig. 72.

Very large mills of 80 stamps or more are often built on the back-to-back principle illustrated in Fig. 51, the two rows of batteries forming practically independent mills. The manner in which the mill is laid out must depend entirely upon local conditions and the nature of the mill site available, it being always remembered that it is of paramount importance to secure a good solid rock foundation upon which to erect so heavy a structure.

Ore-feeders.—Originally batteries were always fed by hand, and even after the introduction of machine feeding the former method was for some time given the preference, as better results were said to be obtained by it. As mechanical ore-feeders have gradually been improved, it has been found that machine feeding is in every way superior to hand feeding. It increases the capacity of the mill at least 15 per cent. by keeping up a steady, uniform supply of ore, and for the same reason the wear and tear of shoes and dies is less, as the die is always kept covered with a bed of ore, so that there is no pounding of metal on metal. There is accordingly also less risk of breaking a stamp stem or of splitting a head. It must be remembered that in the stamp-mill the power is only indirectly applied to crushing, its direct action being merely to lift up the stamps; hence the machine consumes exactly as much power when running empty as when crushing at its fullest capacity, so that an automatic feeder actuated by the mill itself is an economic necessity. In hand feeding too, ore is thrown in by the shovelful, so that the mortar is alternately too full and too empty; when the former condition prevails there is great likelihood of bursting a screen. The feeder in charge of a mill can tell by the sound and feel of the stamp when the battery requires more ore; the

sound of metal striking on metal is very characteristic, and the peculiar hard feel of the blow of the stamp when it is striking on the die or on a very thin layer of quartz only is also easily recognised with a little practice. It is usually one man's duty to feed two five-head batteries, and in small mills he has also very often to spall up the stone to the size required. Of course, hand-feeding at the best is bound to be irregular, whereas with a good mechanical feeder the utmost uniformity is attained if the quartz be delivered to the machine broken down to a small size. It is the object of machine-feeding to keep a thin uniform layer of ore upon the dies, which is being constantly kept at one definite thickness by the gradual supply of ore to the mortar just in proportion as it is required. The power absorbed by machine feeders is so very little that it may be quite neglected. There are very many patterns of machine feeders. A very simple device consists in making the end of the ore shoot from the hopper, movable about a horizontal hinge. By means of a strong spring or a counterpoise it is kept up at such an angle that the ore cannot run down it; attached to the end of this hinged shoot is a bumper rod so placed that it shall be struck by the tappet of the middle stamp whenever there is not sufficient ore under this stamp to keep it off the die. The shoot is thus jerked down, and the ore on it is delivered into the feed shoot of the mortar. This is a simple and inexpensive arrangement, and works fairly well with dry ore broken to a uniform size.

There are three well-known ore-feeders whose principle is somewhat the same, namely the Victor, the Stanford, and the Tulloch ore-feeders. Among these priority in

point of time belongs to the Stanford, which appears to have been the first mechanical feeder invented. Each of these machines consists of a hopper carried upon a light wooden frame, which is usually on wheels so as to allow it to be pushed back out of the way when the mill is being cleaned up or repaired; the hopper is usually about 2 feet 6 inches square at the mouth and about 3 feet high, so that it has a capacity of about 10 cubic feet. The hopper is open at the lower end, and delivers into a tray or shoot so hung that it is capable of a forward movement when the bumper rod connected with it is struck by the tappet of the stamp. At each forward movement a certain portion of quartz, the quantity of which can be regulated by various devices, is pushed forwards and delivered into the feed shoot of the mortar. Any one of these ore-feeders answers very well with dry or clean ore, but if the ore be muddy and sticky, the hopper and tray are apt to become choked, and the machine cannot then be relied on for steady feeding. The Tulloch is perhaps the best machine of the three, but should only be used when the stone to be fed is not very clayey; its main advantages are that it is a cheap machine, simple and not likely to get out of order, and easily repaired when worn out; it is not a heavy machine, weighing 6 cwt. ready for shipment. Fig. 73 gives a general view of this feeder.

Another machine that is sometimes used is the roller ore-feeder. In this the bottom of the hopper is closed by a hollow iron cylinder about 9 inches in diameter; by means of a very ingenious friction grip this roller is revolved whenever the tappet strikes the bumper rod, which forms part of the friction grip, and by the revolution of the roller some of the ore is delivered into

the mortar box. This machine does not, however, work satisfactorily; it will not do at all good work with clayey

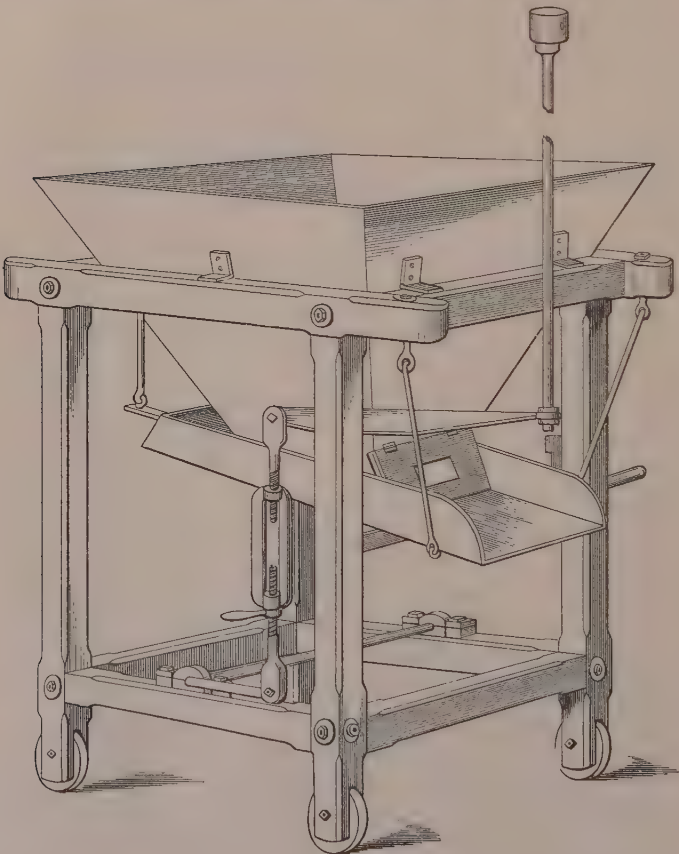


Fig. 73.

ores, and when the ores are dry, especially if at all fine, it is apt to deliver so large a quantity at once as to choke

the mortar. This feeder is accordingly but little used. The best ore-feeder at present on the market, and one which combines all the qualities essential to successful feeding, is Hendy's Challenge ore-feeder. In this machine, shown in Fig. 74, the bottom of the sheet-iron hopper is closed by an inclined cast-iron plate the revolution of which carries the ore into the mortar. By means of bevel gearing this plate is connected with a friction disc, which can be revolved by the action of a kind of friction ratchet; this friction gear is actuated by a lever, attached to a rocking shaft, and from this shaft a bumping rod is carried up to the tappet of the middle stamp. A hole is bored through the guide-block of the middle stamp and the bumping rod passes through this hole; its upper end is protected by a buffer of india-rubber to lessen the shock when it is struck by the tappet. By means of a small hand wheel turning a screw which controls the play of the lever, the position of the bumper can be adjusted and the feed regulated with the utmost nicety. Whenever the depth of ore on the dies falls below the desired amount, the tappet strikes the bumper; this blow is conveyed to the rocker shaft, and thence by means of the lever and grip to the friction disc, which is revolved through a greater or lesser arc; this motion is communicated to the plate, which is in its turn rotated through a greater or lesser angle, and carries forward a proportionate amount of ore into the mortar. This ore-feeder works well even on the stickiest and finest of ores, and if the ore is broken to at all uniform size can be relied upon to feed uniformly. Its disadvantages are higher first cost (about £50) as compared to the others and slightly greater weight, namely, about 8 cwt.; it has moreover a large number of work-

ing parts, which, if injured, are difficult to repair. Upon the whole, however, it is the most reliable ore-feeder as yet introduced, and should be given the pre-

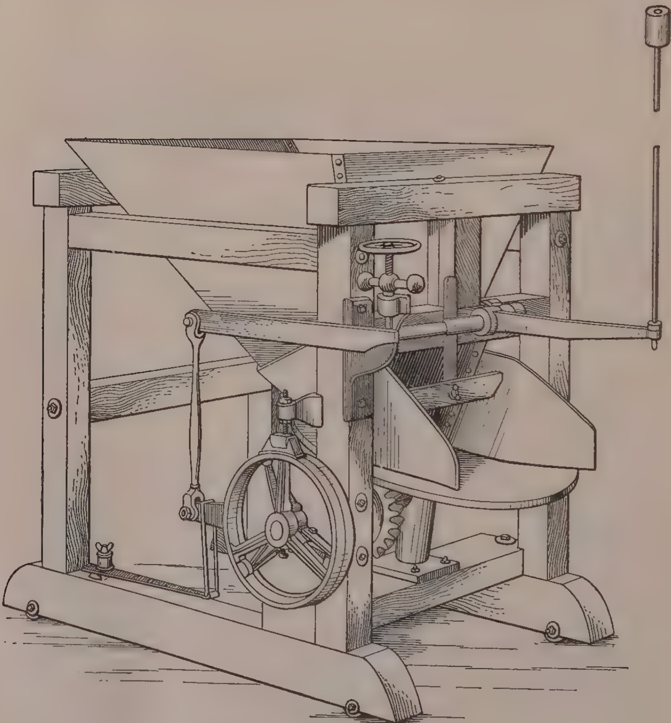


FIG. 74.

ference whenever its higher cost is not an insuperable objection. A very neat modification of this feeder, known as the suspended Challenge ore-feeder, shown in Fig. 75, is now much used. In this the hopper is done away

with, the revolving iron plate and the friction gear driving it being suspended directly from the bottom of the feed

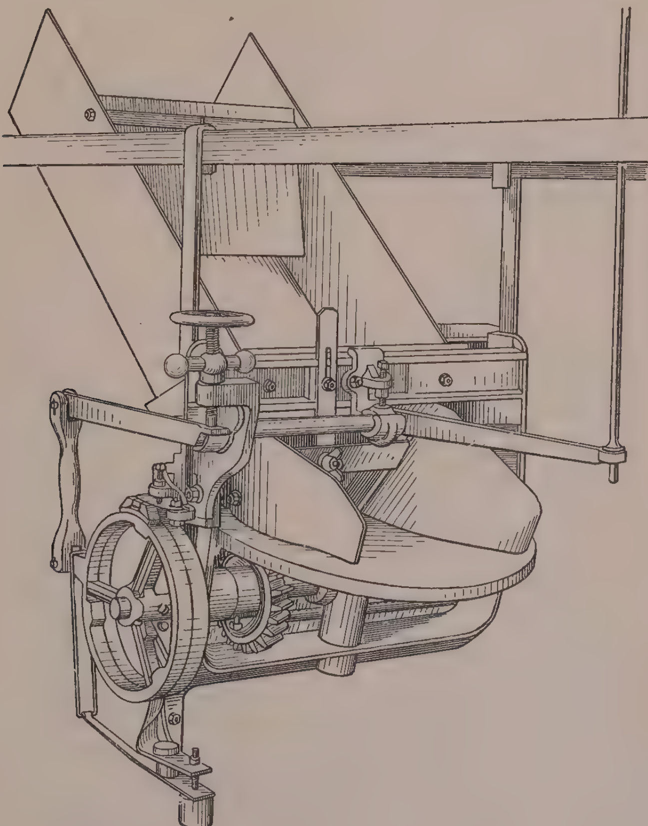


FIG. 75.

shoot that comes from the bins. The action is of course precisely the same as with the older pattern, but the space taken up by the hopper and its frame is economised,

and the back of the batteries rendered more readily accessible. In the newest models, the bumper rod leading to the tappet is done away with, the friction grip lever terminating in a fork which embraces the stamp stem, and which is worked by a collar secured to the latter; the friction gear, instead of being below the revolving cast-iron plate, is above it, and is thus more accessible and less liable to injury. Various modifications of the original Challenge Feeder are now upon the market.

CHAPTER IX

OTHER CRUSHING MACHINERY—ARRASTRA—HUNTINGTON MILL

Classification of Quartz-Crushing Machines.—It has already been pointed out that the stamp-mill is by far the most important of all gold-quartz-crushing machines; probably at least 90 per cent. of all the gold quartz crushed in the world is treated by means of stamps. There are, however, a certain number of other machines, which possess real advantages, that render their employment advisable under certain circumstances, and these machines are worthy of study by the mill man. The number of gold-crushing machines that have been from time to time, and are still being invented is something enormous; scarcely a year passes but that one or two such machines spring into existence heralded by the customary flourish of trumpets, greatly to the benefit, in many cases, of the interested parties, although they mostly sink into well-merited oblivion with at least equal rapidity. No useful purpose can be attained by a description of these failures, and I shall here confine my attention to such machines as have proved of real practical use, and which have succeeded in maintaining their ground, when local circumstances are favourable to their adoption. Nothing

is more certain than that the others will be revived from time to time, with slight modifications and under new names, and that with most of them history will once again repeat itself. Whilst but little harm can result from the trying of new machines upon well-established mines whose existing machinery is sufficient to earn dividends and to recoup the losses that may arise from unsuccessful experiments in other directions, no well-advised mine-owner would think of relying upon machinery that has not thoroughly stood the test of long practice for developing a new mine; the risks incident to mining proper are quite sufficient without multiplying them by those attending the employment of unproved machines.

The various machines (excluding gravitation stamps) used for quartz-crushing may be divided into the following classes:—

1. Machines that grind under the action of heavy weights; to this class belong the arrastra, and the Chilian mill with its modifications, such as the Bryan roller mill, &c. Of these the only machine that can be said to have stood the test of experience is the arrastra. The Chilian mill is used on a large scale in several of the gold-fields in the Urals, though its displacement by stamp-mills is probably only a question of time. It has held its ground there up till now, partly because much of the ore treated is very soft, and partly because the only stamp-mills known there up to the present are defective in almost every detail. One of the best of these Chilian mill plants is to be found at the Ouspenski Mine, Kochgar; there are two mills with pans 15 feet in diameter and 30 inches deep, fitted with 3 rollers, 4 feet in diameter by 1 foot wide; these weigh 3 to 4 tons each, and have steel hoops. The beds are of steel and the front portion of the pan is fitted with

screens of punched sheet-iron with holes about 0·1 inch in diameter; the depth of discharge above the bottom of the pan is nearly 12 inches. The speed of working is 12 to 18 revolutions per minute. Mercury is fed into the pan when the mill is running on oxidised ores: the pulp runs over amalgamated copper plates 12 feet long, then over a shaking table also covered with amalgamated plates, and finally over Embrey vanners. The entire plant treats 40 to 50 tons in 24 hours, and is driven by a 35 H.P. engine, which is not, however, worked to its full power. The ore averages 15 dwt. per ton, and the tailings about 3 dwt.

2. Machines that grind by means of balls or rollers under the influence of centrifugal motion or of gravity, or both combined. Of these there are numerous varieties, such as the Globe mill, the Crawford mill, the Cyclops mill, the Ball mill, &c., in all of which the actual crusher is spherical; and the Niagara pulveriser, the Howland pulveriser, and the Huntington mill, in which the crusher is cylindrical.

These machines have all been grouped together because they are characterised by the fact that the crushing action takes place between one or more rounded crushers and a more or less narrow circular track or bed. None of them can be looked upon as machines of proved value for ordinary milling except the Huntington mill.

In some special cases dry crushing may be employed with advantage, especially in regions where water is scarce, and on ores containing the gold in a state of extremely fine division. Such ores may be dry crushed and then submitted to direct cyanidation. Several machines of this class have given good results when used in this way, more especially the Ball mills made by the

Grusonwerk Company (Fried. Krupp). A convenient size takes about 11 I.H.P. to drive it, crushes about $\frac{1}{2}$ ton of average quartz per hour to a 40 mesh, and costs about £300. The consideration of dry crushing is, however, foreign to the objects of this book.

3. Disintegrators of various kinds, in which crushing is produced by the mutual collision of particles of the ore itself at high velocities. These are all dry crushers, and are quite unsuitable for the purpose of ordinary gold milling.

4. Rolls; these are modifications of the old Cornish rolls, the differences consisting chiefly in alterations of detail so as to adapt the machine for fine crushing. Theoretically, rolls should be one of the most economical forms of fine crushers, as they substitute steady continuous action for the intermittent action of the stamps, and use up power only in proportion as they do effective work; they will probably in time take their place amongst recognised gold-crushing machines, but up to the present they can certainly not be said to have done so. What is required in the gold milling practice of to-day is a wet crusher that shall at the same time be an efficient amalgamator (not necessarily, however, conducting both operations in the same apparatus), and rolls cannot be said to have attained this position. At present there are but few isolated instances of gold mines where crushing is being done by means of rolls. The so-called Krom rolls are among the best forms of crushing rolls; the machine as now constructed is by no means the same as originally patented by Krom, and indeed in some details the supposed improvements in which the Krom rolls differed from their prototype, the Cornish rolls, have been found disadvantageous, and have again been discarded.

As now constructed, the Krom rolls consist of a pair of rollers about 2 feet in diameter and 15 inches wide, driven at from 80 to 100 revolutions per minute in opposite directions by means of open and closed belts respectively. The bearings of these rolls are carefully adjusted, and one of the pair is so supported as to be capable of a certain amount of play, the rolls being forced together by means of powerful springs; they are enclosed in housings, and ore is supplied uniformly by means of an ingenious ore-feeder. Fig. 76 shows the arrangement of a Krom roll plant as arranged by Messrs. Bowes Scott and Western, Limited, of London. The ore is first broken in two rock-breakers set one above the other, from the lower one of which it passes into revolving screens; all that is fine enough goes to the rolls, the coarse stone being carried back to the breakers. A similar revolving screen below the rolls allows only the fine to pass through, all that is too coarse being lifted back into the feeder of the rolls. From the Krom screens, the fine ore passes into a mixer, where it is mixed with a proper amount of water, and thence it passes on to the amalgamating and concentrating machinery. This type of plant is said to be in successful operation at gold mines in Australia, Nicaragua, the Ural Mountains, and Siberia. Nothing, however, is known of its actual working capabilities, or the quality or cost of the work performed by it. Among the advantages claimed for it, as opposed to stamps, are those of economy in first cost and lightness. A plant such as illustrated, capable of crushing 40 tons per day of medium quality quartz, through a mesh 0·024 inch square, costs about £1500 and weighs about 30 tons. The makers state that such a plant will require about 32 I.H.P., of which the rolls alone require only 11 I.H.P. These data would

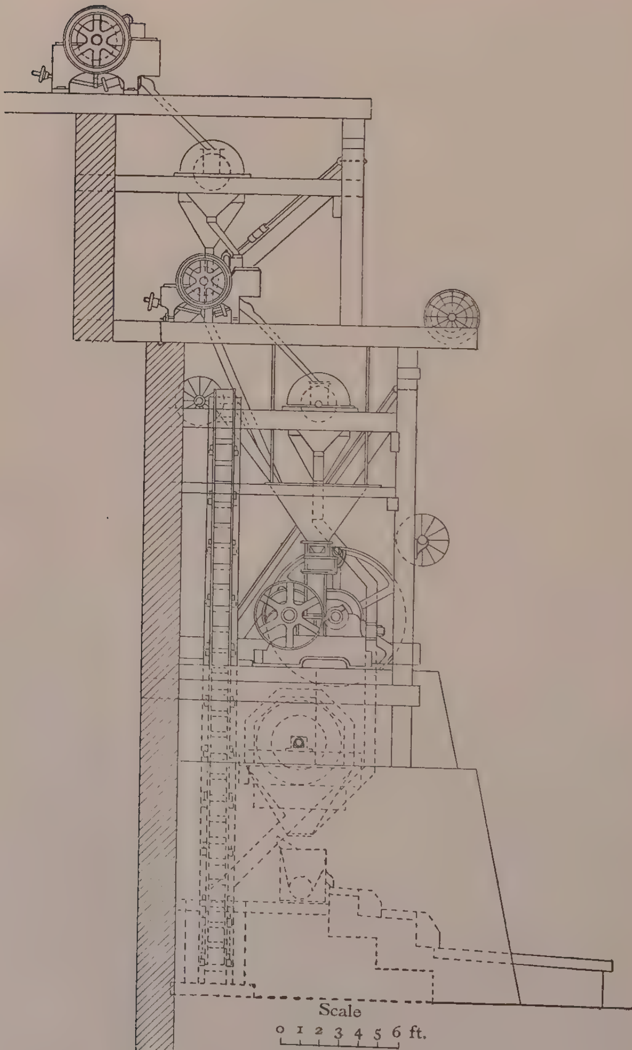


Fig. 76.

make the efficiency of the entire plant about 1·04 cwt. per I.H.P. per hour, or little more than two-thirds of that of the stamp-mill. A very serious objection to the use of rolls as at present constructed is that they have not hitherto been adapted to crush any other than dry ore. The number of mines in the world producing ore so dry that, when crushed, it will pass through a fine screen without clogging is very limited, and there are also very few that can afford either to dry or to calcine their ores, except as a preliminary to direct cyanidation. It would not seem at all difficult to adapt rolls for wet crushing, but until this has been done and the economy of their use, when so applied, has been demonstrated, their sphere of usefulness must of necessity be restricted by the above limitations. Whatever possibilities rolls may possess, these are as yet undeveloped, and it remains for the future to give us an arrangement of crushing rolls thoroughly suitable to the requirements of gold milling.

5. Power Stamps; that is to say, stamps in which the momentum of the stamp due to gravity is increased by mechanical means. To this class belongs the steam stamp, which has been used successfully for the (comparatively coarse) crushing of copper ores, but has apparently proved a failure when applied to gold quartz. For one thing, its proper lubrication presents great difficulties; moreover, although it seems at first sight that the direct application of steam ought to be far more economical than its indirect application through cranks, pulleys, cams, &c., yet this is really not the case; when a gravitation mill is driven by an engine, the most economical form of the latter as regards utilisation of steam can be adopted, whereas in the steam stamp, owing to the necessity of providing a

cushion at either end of the stroke, and to the fact that the cylinder must be of the non-compound type, very much steam is wasted. A modern pattern, the Tremain steam stamp, is stated to have been used successfully at Gunnison, Colorado, as well as at other places. At the first named locality each mortar, 12 inches by 24 inches in area, contains two stamp-heads 9 inches in diameter, and carries screens in three of its sides. Each stamp weighs 300 lbs., but the steam pressure brings its effective blow up to that of a 1,000 lb. stamp; it runs at 200 drops per minute, the length of drops varying from 5 to 8 inches. The crushing capacity of the mill averaged 12 tons per 24 hours on rather soft ore, with a power consumption estimated as equal to 12 H.P. The ore was crushed through a coarse 20 mesh screen (0·035 inch), but much of it was slimed, 48·5 per cent. of the pulp passing through a 100 mesh screen (about 0·005 inch), which portion carried two-thirds of the total values of the ore. The mill appears ill-suited to inside amalgamation, fully 95 per cent. of the amalgam obtained coming from the outside plates. The mill is built by the Gates Iron Works, Chicago. Of course, no steam stamp could in any case compete with a gravitation stamp driven by water-power.

In the Elephant stamp and Dunham's stamp the blow is given by the propulsion at high speed of light stamp-heads, the power being transmitted to the stamp by means of strong springs, so as to protect the machine from injury by the jar of the blow. Neither of these machines have proved successful in steady work, although elephant stamps are sometimes—but very rarely—still used as a prospecting plant. These stamps have a very large number of wearing parts that are rapidly destroyed; more especially are the pins that connect the spring to the

stamp-head exposed to an amount of friction that rapidly cuts them to pieces. In pneumatic stamps, such as the Husband and the Scholl, the principle is the same as in the last-named, except that the spring is replaced by an air-cushion. A very few are at work in places, but they are by no means satisfactory, and have in more than one case been discarded in favour of the gravitation stamp. Husband's pneumatic stamps, manufactured by Messrs. Harvey and Co., of Hayle, Cornwall, have been used in a few isolated instances, in Nicaragua (Central America), and in South Africa, but there are no records available of the results obtained by them. The small size of the mortar compared to the output from it, fitted as it is with screens on three sides, destroys its efficiency as an amalgamating machine. According to the makers its crushing capacity is 40 tons (of tin stuff, not of quartz) crushed from 4-inch cube through a mesh 0·048 inch in diameter with an expenditure of 35 I.H.P. This is equal to an output of 0·95 cwt. per I.H.P. per hour, which is by no means great considering how coarse a mesh is employed. Data are, however, still wanting for an accurate comparison with stamps. The difficulty of proper lubrication seems up to the present to be a vital objection to all forms of power stamp, and the comparatively small size of the mortar is also a disadvantage. A modern machine, which has, however, not as yet been tested in actual work, is Morison's High Speed Stamp; in this the mortar box, stamp-heads, shoes and dies of the ordinary gravitation pattern are employed, but the cam is replaced by a crank moving a cylinder within which is contained a piston attached to the upper end of the stamp stem. Instead of the cylinder being filled entirely with air, as in the last-named machines, the lower part

is filled with wa er, which enters the cylinder from a reservoir attached to it through a small post; in other words, Morison's stamp is a hydro-pneumatic instead of a peumatic stamp. It works very smoothly at a rate of 130 drops per minute, and its crushing capacity seems to be decidedly superior to that of the gravitation stamp; the inventor claims that six heads of his high-speed stamps are equal to ten heads of gravitation stamps, but the crushing efficiency of both patterns seems to be about the same. It remains, however, to be seen how the wear and tear of the new machine compare in practice with those of the cam stamp.

Under this head there may also be mentioned the attempts that have been made to augment the power of the gravitation stamp by making the head drop under the action of some device that shall supplement the force of gravity and cause the stamp to descend at a greater rate and consequently with more momentum than under the action of gravity alone. Such are the stamps, also made by Messrs. Harvey and Co., of Hayle, in which a set of cams above the tappets drive the latter downwards. Other makers have attempted to use springs for the same purpose. It need only be said that all such experiments have proved to be failures even from the point of view of economy of power. And the fact must not be lost sight of in considering all these machines that the modern stamp is an amalgamating no less than a crushing machine, and that the maximum of efficiency in both functions combined is the object to be striven for.

The whole of this part of the subject may be summed up in the remark that it is easy to quote instances where one or other of the above-named machines has been replaced by gravitation stamps, but I know of very few

cases where the gravitation stamp has been discarded in favour of its would-be competitors in gold milling. The only machines that need be described in any detail are the arrastra and the Huntington mill.

The Arrastra.—This, in its simplest form, is a very primitive machine, having been adopted by the pioneers of Californian gold-quartz mining from the practice of the old Mexican silver-miners. It still holds its own in certain places, because it can be built altogether locally and is very cheap to erect, is simple to work, is adapted

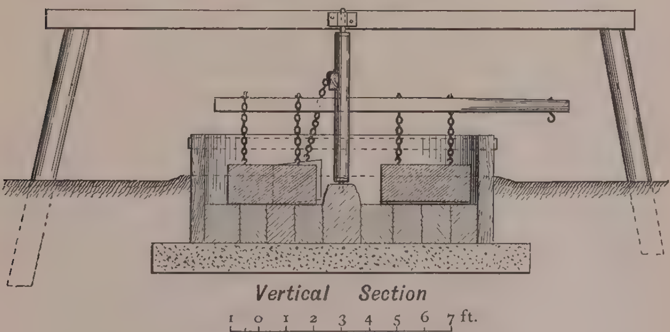


FIG. 77.

to animal power, and is perhaps the most efficient gold-saving machine known. It is therefore especially suitable for the purposes of miners with little or no capital, working their own claims, particularly where the ore is rich and the vein small, and is *par excellence* the machine for the prospector and pioneer in remote and new mining regions. Its use should be confined to free-milling ores, containing a very small proportion of sulphurets, such as compose the out-crops of most quartz reefs. It has also the great advantage that it can be

worked by animal power—horses, mules, or oxen—and can therefore be used in places where fuel is so scarce that steam-power is out of the question, and where there are no streams capable of furnishing water-power. This last consideration is a very important one, and must not be overlooked, as it will sometimes determine the form of crushing machinery that will have to be adopted in any particular case. In its simplest form the arrastra (as shown in section in Fig. 77) consists of a shallow circular bed, which is paved with closely set blocks of hard stone; upon this pavement two or more heavy drags of stone are slowly pulled round, thus crushing the ore under them. The bed is usually laid upon a foundation of a layer of clay beaten down hard and made as firm and solid as possible; this foundation should be two or three feet larger in diameter than the proposed floor of the arrastra proper, so as to project all round it and prevent the escape of mercury; whenever possible this foundation should be made of concrete, and be at least one foot thick. Upon this bed there may be a layer of two or three inches of sand rammed hard, and upon this the pavement may be laid, or the blocks, if well dressed, may be laid directly upon the concrete bed. The outer circumference of the arrastra may be built of blocks of stone, but as the wear upon it is very slight, it may with advantage be made of wooden staves held together by a couple of iron hoops. At times the pavement of the arrastra is well below the surface of the ground, and then the lower part of the walls may simply consist of clay well beaten down. Wooden staves form, however, the most satisfactory wall, the height of which generally varies between 1 foot 6 inches and 4 feet 6 inches. The pavement must consist of blocks of hard stone, carefully dressed and jointed,

and laid as evenly and as closely together as possible. The depth of the pavement is usually between 1 and 3 feet, and the stones between 6×6 and 12×12 inches in area; the larger they are, the better. The joints should be filled with good cement, or failing this with very fine sand, worked in as thoroughly as possible. The best material for the pavement is probably a fine-grained basalt; next to this, a granite or quartzite. If possible, stones should be selected that do not assume a smooth polished surface by wear, but that retain a rough texture, as these are far more efficient as grinders. At the same time the stone should not be markedly softer than the ore to be treated. In the centre of the arrastra is built in a block of stone or of wood, which carries the central revolving post. This block should carry a step, preferably of cast-iron, in which the pivot of the central post can work. To this post are bolted a number of arms varying between two and eight. It is suitably supported at its upper end, and the motive power, whatever this may be, is applied to it either by means of one of the above-mentioned arms or by a separate one. When animal power is used a pole is bolted to it, to which the animals are harnessed by suitable means, or else, as in the figure, the harness is fastened to a prolongation of one of the drag arms. Sometimes the upper part of the post carries bevel gearing, and is driven from a main shaft, which may then drive several arrastras, the main shaft being driven by various motors according to circumstances. A very usual arrangement consists of a horizontal hurdy-gurdy wheel having buckets inclined at an angle of 45 degrees, which is attached by arms to the central post, and which surrounds the outer wall of the arrastra. Water is brought to this wheel by means of a flume,

terminating in a vertical penstock, which delivers the water on to the inclined buckets and thus rotates the central post. The drag-stones are attached to the arms projecting from the central post; they are usually of the same material as the pavement. Their lower face is usually somewhat wider than the upper one, and they are comparatively flat, so as to have no tendency to turn over. Into their upper surface a couple of eye-bolts are secured, rather in front of the median line. These eye-bolts may be leaded in or may be screwed into wooden plugs driven into holes drilled in the drag-stones. They are attached to the arms either by chains or by thongs of raw hide: there is often a provision by means of which the length of these attachments may readily be adjusted. As a rule, the two are of different lengths, so that the front face of the stone when at work forms an angle with the radii of the arrastra. This ensures a thorough turning over and mixing of the pulp. The stone is so hung that the front is just off the bottom, whilst its rear part presses on it. By this means the drag-stone rides over the quartz lying on the pavement instead of pushing it along before it. The number of drag stones is between two and eight, four being the most usual number. They should be as large as can conveniently be put in. Small drags, two or three at times to each arm, should only be used when large stones are not procurable. Their weight varies widely, the limits being two hundredweight and one ton respectively. Six or seven hundredweight is a very usual size. Four such stones can be readily drawn by a pair of mules. The diameter of such an arrastra is mostly between 8 and 20 feet, and the width of the annular bed between 3 and 8 feet. The number of revolutions made is between 6 and 12 per minute, the

lower speed being that of animal, and the higher that of water or steam power.

After the arrastra has been built, it should be run for a few days on charges of barren quartz sand, until the drags and pavement have been worn fairly level, and all crevices are thoroughly filled up. The arrastra is then ready for use. In the outer wall of the arrastra there should be a series of holes at various heights closed by plugs, and these should deliver into a sluice set on a rather flat grade, say one inch to the foot, which may be furnished with amalgamated copper-plates, riffles, or blankets, as circumstances may require. Arrastras of superior design have been constructed, in which the arrastra proper consists of a wrought-iron pan, about 8 feet in diameter, and a foot or more deep. These pans may be lined with stones, but have more usually a bottom of chilled-iron plates. Many old miners maintain that mercury is least apt to flour in an arrastra where stone works on stone, and most apt to between two surfaces of metal. We really know so little about the phenomenon of flouring that it would be unsafe, in the present state of our knowledge, to deny the truth of this statement, although it is difficult to see any intelligible reason for it. The arrastra is essentially a fine grinder and amalgamator, and is not suitable for breaking down the ore, which latter should be broken small before charging, say to about $\frac{1}{2}$-inch or $\frac{3}{4}$-inch mesh. In small establishments, where there are only one or two arrastras driven by animal power, this breaking is usually done by hand. Larger plants driven by power should always include a rock-breaker and, whenever possible, a pair of Cornish rolls for coarse crushing. The ore should be carefully hand-picked, then crushed and stored in bins; if possible, the plant should be so arranged as to allow of

the automatic delivery of the crushed ore into the arrastras. The amount charged in will vary with the size of the arrastra, the nature of the ore, and the fineness to which it is broken in the first instance. It may be calculated to form a layer between 1 inch and 2 inches in depth on the bed of the arrastra. An arrastra 10 feet in diameter will treat about half a ton of ore at a time. This is spread uniformly all over the bed, damped down with water, and the arrastra started. At first the front of the drags may be so arranged as to be about $\frac{3}{4}$ inch above the level of the pavement. After an hour or two, the ore will have been ground fairly fine, when enough water should be added to form a stiff uniform paste. The requisite amount of mercury is then sprinkled all over the surface of the pulp, care being taken to distribute it as regularly as possible. This is perhaps best done by squeezing it slowly through a piece of rather fine canvas which will break it up into globules. The quantity of mercury used should be about three times as much as the weight of gold supposed to be present in the ore. In case of doubt on this point it should be introduced gradually in small quantities, some of the pulp being panned up from time to time in order to examine the character of the amalgam formed; as long as this still shows the shape of the original particles of gold, or is granular, dry, and hard, more mercury is required, until the amalgam assumes about the consistency of putty; it should never be allowed to get softer than this, and is preferably kept a trifle harder. The mercury should be thoroughly pure and amalgamated with a little sodium, precisely as is done in the case of mercury charged into the mortar of a stamp-mill (see page 305). The most important point to pay attention to, in working the arrastra, is the consistency of the pulp; this should be just thin enough to allow the

drags to work easily and smoothly through it, so as to mix it thoroughly, and yet not so thin as to allow the mercury to sink through the mass, but rather to remain suspended in it, in the form of small globules, until these combine with the particles of gold. The progress of the amalgamation can be observed by taking out samples from time to time and panning or "horning" them; it is complete when no more particles of gold can be found on washing. This usually takes from two to five hours according to circumstances. When this stage has been reached, the pulp should be largely diluted with water and the arrastra worked as rapidly as possible. With power arrastras, provision should be made to enable the speed to be increased to about 18 revolutions per minute; with animal traction the stones may be so far raised as to bear on the pavement with one edge only; in this position it needs very much less force to drag them, and their speed may, therefore, be increased. This tends to settle the heavier particles on the bottom of the arrastra, and to suspend the lighter ones throughout the pulp, which should then be run off through the plug holes, commencing with the uppermost one, and continuing downwards till all has been run off. The pulp is discharged through the sluice box; if the gold is very fine, this box should be lined on the bottom with a length of 5 to 8 feet of amalgamated copper plates. If the ore contains sulphurets, blanket strakes should be employed, and in any case there should be a length of ten to 15 feet of sluice boxes fitted with transverse riffles, about $\frac{1}{2}$ inch deep and 1 inch wide. When the plant consists of several arrastras, they may be so arranged as all to discharge into one common sluice. The length of time occupied in working off one charge is usually between 6 and 12 hours, according to circum-

stances. The quantity of ore treated per day is thus variable within very wide limits; it may be averaged at about 30 cwt. A twelve-foot arrastra of the best construction, driven by power at about 12 revolutions per minute, requires about 6 H.P. to drive it, and will, under favourable circumstances, treat 6 tons of ore in 24 hours. As a general rule, after a charge has been worked off, the arrastra is not cleaned out; it will, therefore, retain the whole practically of the amalgam, a large proportion of the sulphurets, and any ore that may not have been ground fine enough to be suspended in the pulp; a new charge is then introduced, and the operation repeated. Every week or two the arrastra is cleaned out, all its contents being removed as completely as possible by means of scoops and stiff brushes; no attempt is, however, made to scrape out the amalgam that may have found its way down into crevices between the blocks of the pavement. The material collected is washed up in a cradle, or, if small in amount, in pans, and the amalgam saved; any sulphurets obtained are put aside for treatment. A complete clean-up generally takes places only when the pavement is worn out and has to be replaced. The stones composing it are then taken up, carefully scraped and cleaned so as to leave no amalgam adhering to them, and the sand upon which they are laid taken up and cradled till the hard bed below is reached; if this consists of clay, the upper part of it may also be scraped off and washed; sometimes the pavement is laid upon several beds of sand of different colours, and these layers are taken up and washed one at a time, until it is found that the lowest limit to which amalgam had penetrated has been reached. A new pavement is then put in as before, and a fresh campaign ommenced. The length of time that a pavement will

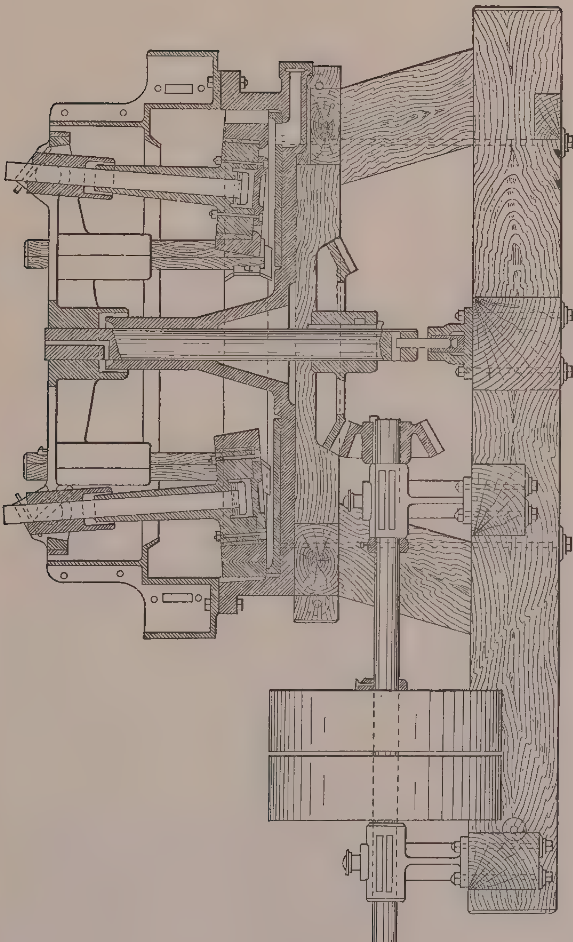
last is exceedingly variable, depending upon a number of circumstances. The labour required for working the arrastra is very little. One man can easily attend to one arrastra on a twelve-hour shift, besides spalling up the ore required for it. A boy is also required for driving the mules or other animals that work it. In the case of a power arrastra, where the ore is crushed by proper breaking machinery, one man can attend to two or three arrastras, or even more if the plant is properly laid out.

On account of the cheapness with which it can be constructed and worked, the arrastra is sometimes used to work up the tailings from stamp-mills, which may contain enough gold to be worked profitably in the former machine, though they may not pay for retreatment in any other way. The use of the arrastra is so entirely dependent upon local circumstances, that all its details are subject to the widest modifications. It has therefore been necessary to describe its construction and mode of action in the most general terms. It will readily be understood that the arrastra, as here described, may be adapted by alteration of its details to suit the circumstances of any given case in which it may be advantageous to employ it.

The Huntington Mill.—This mill has now been in operation for some years in various parts of the world, and has given satisfactory results when working upon ores suitable to it. It is not adapted to very hard ores, though specially suited to clayey ones, and is not, perhaps, as suitable as are stamps to the requirements of large mines, where very big quantities of ore have to be handled. If anything goes wrong with a stamp, it can be hung up without affecting the others until an opportunity arises

for repairing it; moreover owing to the great simplicity of the machine, repairs are usually capable of being executed rapidly and easily, whilst in the case of the Huntington mill any accident usually involves stopping the whole machine. The great disadvantage of the percussive acting stamp-mill, as compared with grinding machines, is that the power required to work it is always the same, whether there is any ore under the stamps or not, whereas a grinding mill runs with far less power when empty than when it is crushing. It is for this reason, among others, that the mechanical ore-feeder must be looked upon as an indispensable portion of the stamp-mill, while its importance to a crushing mill such as the Huntington is far less. In fact, one of the disadvantages of the Huntington mill is the fact that any ore-feeder connected with it is bound to feed at a uniform rate of speed, whatever be the rate of crushing of the machine. It cannot, as in the stamp-mill, be arranged so as to feed ore only when needed, and can hence scarcely be used when ores of very varying character have to be treated; in any case it requires close attention. Compared with the Huntington mill, the stamp has the disadvantages of greater initial cost, greater total weight, and greater time required and cost incurred in its erection. It seems clear, however, that its greater reliability compensates in most cases for all these drawbacks.

The Huntington mill, shown in section in Fig. 78, and in plan in Fig. 79, consists essentially of an iron drum, in the centre of which rises a pillar, which is made to revolve by means of gearing beneath it; the lower portion of the drum is lined with a steel ring, against which the crushing is done. This ring takes most of the wear and tear, and is renewable. Just above this ring,

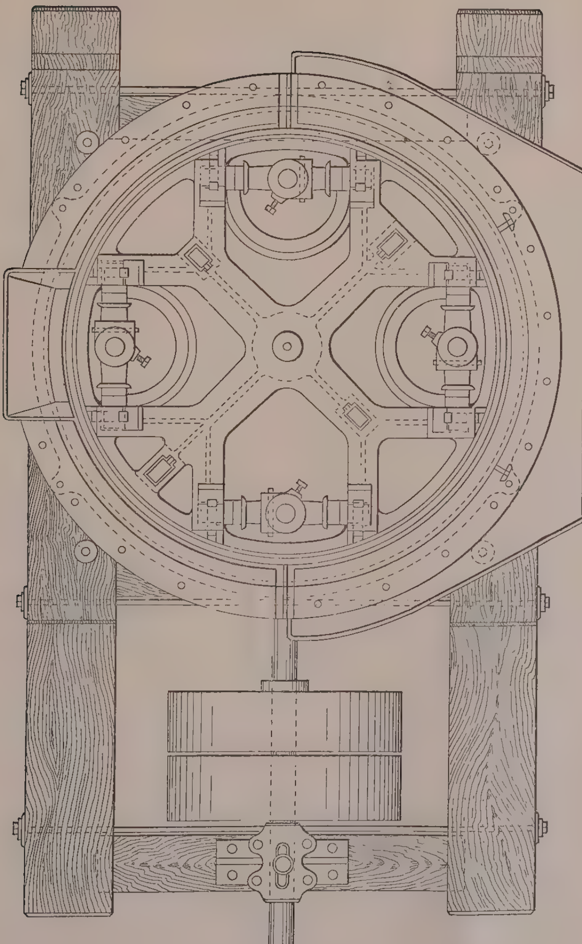


Sectional Elevation

Scale ½" to 1 ft.

FIG. 78.

one-half of the circumference of the drum is occupied by the screens, of which there are three to each machine, their depth being about 9 inches. These screens are, of course, curved to the curvature of the machine, and are held in their places by means of iron screen-frames and keys. The remarks already made with reference to battery screens are equally applicable to those of the Huntington mill; owing, however, to the centrifugal action of the machine, the pulp is thrown against the screens with considerable violence, and they are apt to wear out more rapidly than in the case of the stamp-mill; any kinds of screens may be and are used. The ring of the Huntington corresponds to the dies of the gravitation mill. Crushing is performed by the action of three or four rollers, which are suspended by means of yokes from arms attached to the revolving spindle, the whole of these arms being in one piece, which forms a disc-shaped cover to the pan. The points of suspension of the rollers are a little further from the centres of the vertical spindle than are the centres of the rollers themselves; consequently the rollers, which swing freely in a radial direction, are pressed outwards by their own weight against the ring die, thus aiding centrifugal action. The rollers have renewable steel shells, held in place by wooden wedges, and are capable of revolving freely on the shafts that carry them, there being a provision for the lubrication of the shaft where the roller works on it, whilst at the same time the lubricant is prevented from finding its way into the pully. The disc that carries the roller yokes also carries a set of scrapers, in order to turn over the pulp and drive it towards the ring die. The rollers, like the ring die, are made of steel, so as to minimise their wear, these pieces being, like the shoe and die of the gravitation mill, the



Plan

Scale 1½" to 1 ft.

Fig. 79.

wearing parts of the machine. The rollers are so suspended as to clear the bottom of the pan by half an inch, mercury and amalgam being allowed to accumulate in the space so formed. The iron casing is in three pieces: a bottom pan with central cone through which the driving spindle projects, and the upper part of the drum, which is made in two halves bolted together. The machine is usually set upon a substantial wooden frame. The crushed ore is delivered through the screens to a circular iron lip surrounding them, whence it is discharged on to the tables or other amalgamating apparatus that may be used. Mercury is also charged into the machine just as in the corresponding case when inside amalgamation is employed in the mortar box of the stamp-mill, and the same precautions must be observed with regard to this mercury as in the analogous case of the stamp-mill. It is in order to prevent this mercury from flouring that the rollers are so suspended as not to touch the bottom of the pan.

The mill is made in three sizes as follows:—

Size.	Weight. Tons	cwt.	Revolutions of spindle per minute.
$3\frac{1}{2}$ ft. diam.	3	11	90
5 ,, ,,	6	14	70
6 ,, ,,	8	18	55

Heavier patterns are also made now, especially in the 6-foot size, and some of the modern mills have the screens extending further round the pan, there being five instead of only three. The velocities given above are those recommended by the makers, and are suitable for comparatively soft material. With hard quartz a somewhat higher rate of speed is to be preferred; as the crushing is done by centrifugal action, the speed at which the mill is driven completely controls the power which

it is able to exert. Authentic data are still wanted in order to enable the crushing capacity of the mill to be judged of by the only reliable test—namely the number of tons crushed to a given size per indicated horse-power used per hour. This may be taken at about 1 cwt. per I. H. P. per hour through a screen of 0·024 inch mesh, or little more than two-thirds of the duty that can be obtained from a good stamp-mill. The power required to drive a 5-foot mill seems to be about 10 I.H.P., a $3\frac{1}{2}$-foot mill requiring about one-third less, and a 6-foot mill about one third more power. The actual crushing capacity of a 5-foot mill, which seems to be the size most used, varies between 10 tons and 30 tons per 24 hours, according to the hardness of the materials and the size of the mesh. Quartz of medium softness crushed to a 30 mesh (say 0·024 inch), which may be taken at about the average, can be put through at the rate of about 15 to 20 tons per 24 hours.

In working the Huntington mill, the pan is first charged with a sufficient quantity of clean pure mercury to amalgamate all the gold that may be liberated in it till it is cleaned up, which period may vary from a week to a month. The usual mercury charge is from 30 to 60 lbs. Ore is then fed in steadily, either by hand or by any of the machine feeders used for the stamp-mill, which may be driven by means of cams or by a small shaft taking its power from the line shaft of the mill; the Challenge ore-feeder is well adapted for this purpose. The ore should be broken down by a rock-breaker to $\frac{3}{4}$-inch cube at the outside, before it is fed into the mill. For a large plant of Huntington mills it may be advisable to use two rock-breakers, the first for coarse, the second for fine breaking. Care must be taken that there is no chance

of over-feeding the machine; if anything, it is preferable that the ore should be fed too slowly rather than too fast. When the ore-feeder is once adjusted it may be trusted to feed uniformly as long as the quality of the ore is unchanged. As machine feeding is controlled by the speed of the machine and not by its requirements, as in the stamp-mill, the ore-feeder will have to be readjusted with each variation in the quality of the ore. Feeding the Huntington mill requires, therefore, constant care and watchfulness, and is the most important factor in successful working. The central portion of the pan, when all is working well, should be practically free from ore, its accumulation there being a sign of over-feeding. The water supply needs careful attention and regulation, the pulp requiring to be a good deal thicker than in the stamp-mill; since the pulp is driven through the screens by centrifugal action, it does not require so strong a stream of water to keep the latter clean. Once outside the screens on the lip, the pulp may be diluted to any desired consistency. As it is difficult to see whether all is going well whilst the mill is running, it should be stopped once or twice per shift for examination. Care should be taken that the oil-holes of the central spindle and the roller shaft are thoroughly plugged, whilst, nevertheless, keeping both shafts well lubricated. The cost of crushing in the Huntington mill naturally varies a good deal in different places, according to circumstances. The largest item of wear and tear is for renewals of the rollers and ring die: the amount of this is variously stated at between 2*d.* and 10*d.* per ton of ore crushed, but it is obvious that the hardness or softness of the ore is the essential factor in determining this. At the Brilliant and St. George Mill, Charters Towers, Queensland, there

are six 5-foot mills making 65 revolutions per minute and crushing at the rate of 20 tons per day each. The chief items of cost are as follows per ton of quartz crushed :—

	s.	d.
Wages	3	4·14
Fuel	2	0·61
Renewals and repairs . . .	2	6·05
Stores	1	3·39
Miscellaneous	0	2·32
Total cost per ton . .	9	4·51

In the often quoted instance of the Spanish Mine, Nevada County, California, circumstances are especially favourable to the employment of the Huntington mill, the ore being practically a decomposed soft clayey slate with quartz stringers ; the total cost of milling by water-power (which has, however, to be purchased, amounting to about 15 per cent. of the entire cost) is about 12*d.* per ton of ore ; of this amount 2·3*d.* is the cost of renewing roller-heads and ring dies, and 0·6*d.* that of the screens. In this mine, mercury is charged into the mill at the rate of $\frac{1}{4}$-oz. per ton of ore crushed in addition to the quantity charged at the commencement of the run ; the loss of mercury is very heavy, amounting to 0·25*d.* per ton. The mill is cleaned up every month, there being an arrangement which admits of the quicksilver and amalgam being conveniently discharged from the bottom of the pan, when cleaning up the machine.

CHAPTER X

AMALGAMATION—INSIDE PLATES—COPPER-TABLES—MERCURY WELLS—AMALGAM TRAPS—LOSS OF MERCURY

Gold Extraction.—After the gold quartz has been crushed to the requisite degree of fineness by means of the machinery already described, the process of extraction of the gold from it commences. It must be understood that in practice the two operations go on simultaneously, but, for the sake of clearness, I have considered it advisable to treat the two branches separately, so as to distinguish between the mechanical process of ore crushing, and the chemical process of gold extraction. It has already been pointed out (page 17) that gold occurs in its ores in various forms, which may be separated into two classes, free or amalgamable gold, and non-amalgamable gold, which has to be extracted by means of special methods. Whenever an ore contains free gold in appreciable quantity, this is always extracted by means of amalgamation. Amalgamation may take place either inside or outside the mortar box (or other crushing machine); when outside amalgamation is practised, the apparatus used consists either of amalgamated copper-tables, or of mercury wells, or both. A number of machines have been introduced from time to time under such titles as gold savers, amalgamators, &c.,

but not one of them has come into extended practical use, with the exception perhaps of Laszlo amalgamators and Hungarian mills, both of which are used in Hungary; both consist essentially of shallow iron pans about 2 feet in diameter and 4 to 6 inches deep; the bottom is covered with a layer of mercury, and inside them there revolves a muller (of iron in the former and of wood in the latter machine), by means of which the stream of pulp is forced into close contact with the mercury. These mills are worked in pairs, a pair treating up to 3 tons of ore per 24 hours; a high efficiency is claimed for them as amalgamating ap pliances. None of the other machines deserve mention. Amalgamation, or the combination of gold with mercury, is a chemical operation, which proceeds by surface contact, and therefore demands a certain time for its completion; all that is needed to insure its completeness is that clean gold shall be in contact with a surface of clean mercury for a sufficiently long period for thorough combination to take place. Whenever these conditions are realised, complete amalgamation of all free gold will result.

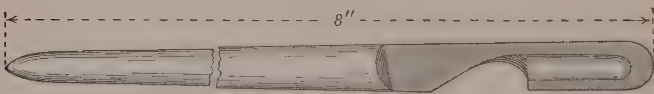
Inside Amalgamation.—As amalgam is less apt to be floured than unamalgamated gold, seeing that it cannot be rendered brittle by percussion, as it moreover has a certain amount of plasticity, and as it is three times as heavy as the gold which it carries, so that the loss of a given weight of it entails a loss in value only one-third as great as the loss of an equal weight of gold, it is advisable to convert gold into amalgam at the earliest possible moment in order to minimise the loss of the precious metal. In its simplest form, inside amalgamation is practised by charging regularly small quantities of

mercury into the mortar box. As a rule mercury is charged once every two or three hours. The quantity used must be about three times as much as that of the gold supposed to exist in the ore crushed in the interval. The mercury falling into the mortar is broken into small globules by the churning action of the stamps and the wash of the pulp, and becomes suspended in and thoroughly mixed with the latter. There it comes in contact with the particles of gold, and amalgamates them more or less thoroughly. Part of the amalgam thus produced gradually coheres into small lumps which sink down and settle in between the dies; some is projected towards the screens and passes through them, to be caught by the outside appliances, or else, when inside copper-plates are used, it may adhere in part to these. Whether there are or not inside plates, the process of inside amalgamation is precisely the same. Inside plates, however, facilitate the collection of the amalgam, and help to retain it in the mortar. They also assist the process indirectly, because a mortar fitted with an inside plate below the screen is necessarily wider, and must have a greater depth of discharge than a plain mortar, and the ore, thus retained for a longer time within the mortar, is more completely amalgamated by being kept in contact with mercury during a longer period. The methods of securing and attaching inside plates have already been detailed (page 150). In the Alaska Treadwell and other patterns of mortar, riffles are used success fully instead of copper-plates for collecting amalgam. The copper plates should not be less than $\frac{1}{4}$ inch thick, and must be carefully amalgamated. They are best silver-plated, and in fact are treated exactly as will presently be described under the head of outside copper-plates.

Unless the ore is very rich, these plates are only cleaned up once a week. If this has to be done more frequently it is advisable to keep duplicate chock blocks with copper-plates attached. The block in use with its plate is withdrawn, and a fresh one substituted without loss of time, whilst the plate which has been withdrawn can be scraped and cleaned at leisure. In a large mill only one spare chock block and plate need be kept, if all are, as they should be, strictly interchangeable, for in that case the plate from the first box can, after cleaning, be set to work in the second box, and this one in turn in the third, and so on. The high cost of copper-plates renders this mode of procedure advisable.

The condition of the escaping amalgam, as caught on the first outside copper-plate, gives valuable indications as to the progress of the amalgamation, and especially as to whether the proper amount of mercury is being charged into the box. If this amalgam is hard, brittle, granular, or inclined to crumble under the fingers, there is not enough mercury; if it is thin, soft, and pasty, there is too much, and the supply needs regulating accordingly. The amalgam when it settles on the plates outside the battery, should do so in moderately firm, coherent crusts harder when the gold is very fine, softer when it is coarse; it should not be pasty, but in such a state that it yields readily to the pressure of the finger and retains the impression. Machines for automatically feeding any desired amount of mercury into the battery have been introduced, but are very rarely used, most mill men preferring to feed it by hand, as the quantity can thus be better regulated to the requirements of the battery. A "spoon" is used for measuring the charge of mercury; it is here shown in half size (Fig. 80); the cavity shown

($\frac{1}{4}$ inch in diameter and $1\frac{1}{4}$ inches long) will hold about $\frac{1}{2}$ oz. of mercury. The spoon is usually cut out of a piece of sound, dry, hard wood, the cavity being burned out by means of a red hot wire; the size of the cavity may readily be varied, as found necessary, whilst the frequency with which the charge is put in forms a further means of regulation. Mercury should be kept in the mill in closely-stoppered bottles of very thick glass, of such a shape that they can be firmly held in the hand. Each bottle ought to hold from 3 to 5 lbs. of mercury. The mercury should be kept in the mercury-room under a layer of mercuric nitrate in a tubulated receiver (page 65); from this the bottle should be filled, and a little



Scale, 6″= 1′

FIG. 80.

sodium amalgam added. The sodium amalgam may be kept in stock, or else be freshly prepared as wanted, the latter being the better method. A little mercury is heated gently in a porcelain basin or small flask, and then a few chips of clean, dry sodium thrown in, and if necessary pressed below the surface of the mercury with a glass-rod. The amalgam so made is then poured into the mercury bottle, which is filled up with mercury and stoppered. The sodium should be used in the proportion of a piece about the size of a pea to each pound of mercury. The usual test for the proper quantity is that the prepared mercury will just commence to amalgamate iron. A cut nail that has been filed bright may be

immersed in the prepared mercury; if it amalgamates freely, there is too much sodium; if the mercury does not adhere to it at all, too little. The mercury should just adhere slightly to the edges of the nail, when the proportion of mercury may be taken as correct. It must not be forgotten that the efficiency of sodium amalgam depends entirely upon its being kept from contact with air, as it is decomposed by moisture. The complaint is often heard that sodium does not save mercury from sickening in the mill; but it will generally be found, on investigation, that such complaints arise from mill men keeping their sodium amalgam in the open air, or even at times under water, or solutions of various chemicals! These men may certainly have taken sodium amalgam to begin with, but this will very soon have lost its sodium under such treatment, so that they were, in fact, not working with sodium amalgam at all. If flouring is to be prevented, the prepared mercury must be kept in well-stoppered bottles. The tare of these bottles should be known, and when filled with mercury, each bottle should be weighed, and its weight registered in a special mercury-book kept for the purpose. A check on the daily consumption of mercury in the mill is thus obtained.

Cadmium amalgam is stated to have been used with success in California in the place of sodium amalgam,[1] but no data are quoted to substantiate this statement. Inside amalgamation is only unsuitable in the case of ores containing by far the larger proportion of their gold locked up in sulphurets, especially when much antimony or arsenic is present. If, however, care is taken to keep the mercury pure and well charged with

[1] Eleventh census of the United States, 1889.

sodium, but little loss from flouring need be apprehended in any case. When the proportion of free gold is very small, it may not be worth while to use inside amalgamation. With very poor ores it is sometimes a more important consideration to put a very large quantity through the battery than to save gold very closely; in that case there should be no inside plates, and the depth of discharge should be kept low. Nevertheless, mercury may with advantage be charged into the battery, because a certain amount of amalgam will always be there retained, and, moreover, this forms the most convenient mode of keeping the outside plates uniformly supplied with mercury.

Outside Amalgamation.—This is now mostly carried on by means of amalgamated copper-tables. These consist of a sheet of copper, the upper surface of which has been thoroughly amalgamated; to the copper amalgam so formed, a thin film of mercury will always adhere, and this film immediately amalgamates any gold that may come into contact with it, the gold amalgam so produced adhering to the surface of the copper amalgam unless there is a great excess of mercury present. When in good working order the amalgamated table will thus at once retain any particle of gold in the pulp streaming over it, that may come into contact with it. The theory of table-amalgamation, therefore, demands a *clean* surface of copper amalgam, carrying a *small* excess of mercury, with which every particle of the escaping pulp shall be brought into contact.

The lip of the mortar-box discharges on to an upper table sometimes called also an apron, and from this on to a second table, although this practice is distinctly bad. Sometimes a lip-plate is attached to the battery itself,

and sometimes an arrangement is adopted which consists of a set of narrow plates, sloping alternately towards and from the screens in a zig-zag manner. There is no object to be gained by these arrangements that cannot be attained equally well by lengthening the apron, unless economy of space is an object, and the arrangement is then adopted merely so as to occupy less room. If possible, however, lip-plates should be avoided. The best arrangement consists of a solid table on independent firm foundations, which are so arranged that its inclination can readily be adjusted. This may be done either by supporting the lower end of the table upon folding wedges which are carried on solidly-constructed trestles, or else by means of bolts which pass through nuts secured to the trestles, and upon the heads of which bolts the tables then rest; the latter arrangement is perhaps the better one, but, in either case, provision should be made to prevent the tables, once adjusted, from being displaced by the jar of the battery. The upper ends of the table should be hinged to a strongly built trestle or are sometimes hung by iron rods from a portion of the structure independent of the floor and of the frame of the stamps so as not to be affected by the jarring of the machinery. This is important, as irregular jarring prevents the amalgam from adhering properly. The case is, however, far otherwise with a copper-table that receives a regular uniform of vibrating or shaking motion. Such tables have been employed with a longitudinal and a transverse shake. The latter has been used in California in the form of a table 4 feet square vibrating backwards and forwards at right angles to the floor of the pulp at the rate of about 200 1-inch strokes per minute; such a table needs only a very flat grade, not exceeding half-an-

inch to the foot, and is said to be a very efficient gold-saver when placed below the usual fixed copper-tables. The tables should have sides about 6 inches high, and the copper-plates should exactly fit into their places and be secured by means of cleats or wedges. The copper-plates should not be either screwed or nailed down to the tables. There are two modes of arrangement in vogue, each of which has its advocates; in the one, the table is continuous, the plates fitting each other accurately and being sometimes brazed together into one sheet. In the other, the table is broken up into steps, there being a drop of from 2 to 6 inches at each step. When the latter arrangement is adopted, it is found that nearly all the gold is caught at the steps where the splash from the successive upper plates strikes the lower ones. When a layer of pulp is slowly streamed over a copper-table the particles of gold tend, by virtue of their superior specific gravity, to settle down upon the copper. The force with which a particle tends to settle varies directly as its mass, but inversely as its superficial area; in sinking under the action of gravity, work has to be done in displacing particles of water and in overcoming their cohesion, and the greater the surface of a body the more particles of water are there to displace, and the greater the force required to be exerted before the body can sink. As this force depends on the mass of the body, there will be a certain point, as its surface increases, at which the work to be done in displacing the particles of water will be greater than the force of gravity which is acting upon it, and in this case the body will be unable to sink, although it will have even less tendency to rise in the water. The relation of volume to surface (and therefore of mass to surface for any given substance) is greatest when the shape of the

body is spherical and least when it is an infinitely thin sheet; and as a sheet must always tend to assume a horizontal position in a stream of water, the resistance to sinking is greatest for this form of body. The very great malleability of gold admits of its being beaten into excessively thin scales, and therefore there must always be in the pulp issuing from a stamp mill, a certain quantity of gold which, in spite of its high specific gravity, has no tendency to sink. When the pulp falls over a drop in the plates, these particles are more likely to be brought into contact with the table and thus amalgamated, than if the stream were allowed to flow on uniformly, without interruption, and for this reason it seems that stepped tables should be preferred to plain ones. The length of the copper-table depends on many considerations, the most important one being the nature of the gold in the ore. If the gold is coarse a short table will suffice, but if fine a proportionately longer one must be used. The length varies accordingly between 5 and 20 feet. A good average length is 15 feet, or say five plates 3 feet long, with a drop of about 2 inches in depth between each. The grade of the plates must be determined by the character of the ore and the supply of water. Plates must always be so adjusted as to be accurately horizontal at right angles to their length, so that the depth of pulp flowing over them is uniform throughout; this is an important consideration to which much care and attention must be devoted, as uneven distribution of pulp means a loss of efficiency in the action of the table, seeing that one side would be carrying too much and the other too little pulp. It need scarcely be said that no portion of the surface of the tables may ever be allowed to run dry; great loss of gold would be the immediate result, as dry

particles of gold or amalgam readily float on water by reason of the film of air adhering to them. The grade of the plates and the water supply must be so adjusted to each other that each particle of the pulp may travel slowly and steadily down the plate, rolling over and over as it does so. It is better to work with a small water supply and a high inclination of table than under the opposite conditions, but there should always be ample water to make a thinly fluid pulp. Finely-crushed ore will require more water and less grade than coarse-crushed; an ore rich in heavy sulphurets—especially galena—will require both more water and a heavier grade than a clean quartzose ore, whilst a clayey ore requires more water and less grade. The water supply must be such that the sand can nowhere settle upon the plates, but must always be kept in motion in regular waves across the table. More water should not, however, be used than is necessary just to keep the plates free from any accumulation of sand. Too much care and attention cannot be given to this most important detail; the due adjustment of the grade of tables and the quantity of water determines to a great extent the efficiency of the mill as an amalgamating machine. The grade of plates varies between $\frac{1}{2}$ inch and 2 inches to the foot, 1 inch being about an average grade. Stepped plates require rather less grade than plain ones, as the velocity acquired by the pulp in its drop tends to carry it forward. The drop should never be so great as to scour the plate upon which the pulp falls; about 2 inches is a fair depth of drop.

Amalgamating the Plates.—The copper-plates should be $\frac{1}{8}$ inch thick and of the purest copper procurable. The best brands of Lake Superior or electrotype copper

should alone be used; any impurities will cause local galvanic action, corrosion of the plates, and a foul surface on the amalgam. The plates should be carefully rolled, so that their surface may be as true as possible, and should be delivered to the mill in thoroughly good order in this respect. They are first annealed, which is usually done by heating them over a fire of shavings and small billets till they are hot enough to char a piece of paper laid on them. During this heating the copper-plates may be supported on a sheet of iron to keep them from buckling. If at all buckled when cold, they must be flattened by gentle blows from a hammer, a piece of wood being interposed between the plate and the hammer. If they require very much hammering they ought to be annealed again, but this is scarcely ever necessary. The object of this annealing is to allow the molecules of copper to assume their normal distance apart, and thus render the absorption of mercury more easy and regular. During the heating, the face of the plate which it is intended to amalgamate should be kept uppermost. Fine sand (sea sand if obtainable) is then sprinkled on the upper face of the plate, well moistened and rubbed in with a block of wood until every portion of oxide is removed and the plate has a uniform red surface, care being at the same time taken not to scratch it. The sand is then washed off, and the plate dried and polished with fine emery paper folded over a block of wood. A perfectly clean dry surface is thus produced. A mixture is then made of about 10 parts of sand to 1 of coarsely pounded sal-ammoniac; this mixture is damped with water, and clean pure mercury is sprinkled into it by squeezing through canvas. This mixture is then rubbed over the plate with a piece of canvas or blanket, when

amalgamation will at once commence; more mercury must be sprinkled on the plate from time to time, and the rubbing continued until a uniformly bright silver surface is obtained. As an approximate guide to the quantity of mercury required, it may be mentioned that each square foot of copper will retain $\frac{1}{2}$ oz. of mercury, this being the actual amount which I found in some experiments on the subject. The effect of the sal-ammoniac is to dissolve off any film of oxide of copper that may have formed on the surface of the copper so as to promote metallic contact between the mercury and the copper.

The amalgamated plate is next well washed with water and kept till the following day; it will then probably be found that the plate is dulled and covered with a coating of a greenish-grey substance; in very thin films this substance may show different colours due to interference, but when sufficiently thick it is always greenish-grey. This substance invariably forms on the surface of amalgamated copper-plates, and is due to the oxidation of part of the copper of the amalgam. I have analysed it and find it to be a hydrated oxide of copper, with sometimes some carbonate; probably, when water containing sulphates is used, a basic sulphate may also form. It is soluble in a number of substances, such as dilute acids, ammonia, and potassic cyanide. Usually the plate is rubbed up with a dilute solution of cyanide, a little more mercury being at the same time rubbed in. It is, however, better to prevent the formation of this troublesome coating by replacing the copper in the amalgam adhering to the table by silver. This may be done by rubbing in silver amalgam instead of mercury at this stage. This silver amalgam may be

prepared in various ways, but best, perhaps, as follows: —A sufficient quantity of silver coin (about $\frac{1}{4}$ oz. per square foot of surface of the tables) is dissolved in dilute nitric acid in a porcelain basin with the aid of a gentle heat. The solution is evaporated to dryness very gently, preferably over a water-bath, and then heated till the saline mass commences to fuse, and till all its bluish tinge is turned to greyish-black, this change indicating that all the soluble cupric nitrate is decomposed, insoluble cupric oxide being left behind. The salt is then dissolved in a small quantity of distilled water and filtered into a jar or beaker. Pure mercury, to the weight of about three times that of the silver used, is poured in, a few drops of nitric acid added, and a few pieces of bright iron floated on the surface of the mercury. The silver will at once commence to precipitate and be absorbed by the mercury forming silver amalgam, the process taking a few days to complete thoroughly. The silver amalgam so produced should be cf a pasty consistency. This amalgam is then rubbed hard all over the surface of the amalgamated plate, which is kept moist with a dilute solution of potassic cyanide; a good rubber for this purpose is made from a strip of pure indiarubber, $\frac{3}{8}$ inch thick and about 6 inches long screwed to a strip of wood, which forms its handle, so as to project $\frac{3}{4}$ inch. The rubbing must be continued until the whole of the plate is completely coated with silver amalgam, which will then keep the plate from tarnishing.

A still better method consists in employing electro-silvered copper-plates, which are afterwards amalgamated. Such plates can be obtained from any makers of mining machinery, or, if preferred, they can be silvered at the mill. This latter plan should not be adopted unless in

the case of a very large mill, where plates are continually requiring re-silvering; otherwise it is cheaper to buy them ready silvered. Electro-silvered plates are best amalgamated by rubbing with mercury containing a little sodium amalgam, the plate being kept wet; of course silvered plates require no preliminary polishing, and ought to have been annealed before being silvered. Electro-silvered plates usually carry from 1 to 2 ounces of silver per square foot. Sometimes in preparing new plates, gold amalgam is used instead of silver, but this is rarely done except in mills already in operation where there is a supply of gold amalgam on hand. Silver is quite as effective as gold for keeping plates bright, though less so for catching gold; but one or the other should be used in order to keep the surface of the amalgamated plates bright, until a film of gold amalgam has been deposited all over them by the operation of milling. It must not be forgotten that the first bullion obtained from electro-silvered plates will be of a lower grade of fineness than the gold contained in the ore, because it will contain some of the silver removed from the plates.

Various alloys have from time to time been used to replace copper, but none of them have proved successful. The only partial exception may be found in the use of Muntz metal, an alloy of zinc and copper, used for the sheathing of wooden ships; this was employed some twenty years ago in the Thames district of New Zealand, because copper-plates were not obtainable. It is best amalgamated in the same way as copper, replacing the sal-ammoniac by dilute sulphuric acid; Muntz metal is said not to absorb mercury like copper, so that amalgam adheres less firmly to the former than to the latter, whilst the amalgamated alloy is said to be less liable to become

coated and foul. There is always risk of brittle gold owing to some of the zinc of the Muntz metal finding its way into the final ingots of bullion. No advantage has been proved to result from its use, and its employment has never spread beyond the district in which it was introduced by local necessities. Silver plates have been used and answer admirably, the only objection to them being their high cost.

Working of Copper-Plates.—When the plates have been duly amalgamated and placed in position, it is advisible first of all to run some barren quartz through the mill and over the plates. This gives an opportunity of adjusting the water supply to suit the grade of the tables. As soon as all is in order, the mill should be stopped and the plates rubbed up with a piece of india-rubber; if electro-silvered plates are used, a little mercury only need be sprinkled over them till the surface just feels soft; a plate that feels hard has too little mercury, but this latter should never be added in such quantity as to produce any tendency towards forming distinct drops; whenever separate globules of mercury form, it is a sign that too much mercury is being used. When inside amalgamation is employed no mercury need be added outside the mortar. Sufficient will always be carried through with the pulp to keep the plates in order, and, as already mentioned, the condition of the amalgam on the plate nearest the mortar is the best criterion by which the feed of mercury can be regulated. Some mill men prefer to add a part of their mercury inside the boxes, and to sprinkle part over the outside plates. There appears to be not the least objection to this practice, but on the other hand there does not appear to be any advantage in it. Mills run side by side using both methods seem to have given about the same results as regards

saving of gold. With new plates the mill should be stopped every four or six hours, and, when the plates have become "set," once every shift, for "rubbing up," and, if necessary, for taking off the amalgam, although this is usually done at most once in twenty-four hours. For rubbing up the plates, the mill is stopped, the water being allowed to run till all the pulp is washed off the tables; if necessary, the hose-pipe may be used for this purpose. If the plate is discoloured by the formation of a film of the grey deposit, this must first be removed, the solvent mostly employed being a solution of cyanide of potassium; some mill men make a practice of throwing a few pieces of cyanide into the mortar from time to time in order to keep their plates bright. This, of course, is only necessary where non-silvered copper-tables are used, and before these have become properly "set" by the gold amalgam adhering to them. The use of cyanide of potassium was at one time very extended, but of late years it has been largely discontinued, and rightly so, seeing that such a solution of cyanide is a ready solvent of gold, which may thus be carried away and lost. Moreover, electro-silvered plates, which are now being very largely used, do not require the use of cyanide solution, which should indeed never be applied to an electro-silvered plate. The best solvent for the grey deposit is very dilute acid, (not, however, nitric acid), whenever this can be obtained. I have kept new plates beautifully clean by means of a narrow leaden trough fixed just above the top plate; this was kept filled with moderately dilute sulphuric acid and fitted with a syphon formed from a strip of blanket, so as to allow a continuous supply of acid to drip on to the plate. The acid acted, no doubt, not only by its direct solvent action on the deposit, but it also, by attacking the fine

particles of steel (from the shoes and dies) suspended in the pulp, produced a series of electric couples with the copper-plate, which would tend to keep the surface of the amalgam bright. As a rule it is scarcely possible to keep new non-silvered plates bright at first, hence these must be rubbed up very frequently. After the film has been removed, a little clean mercury charged as usual with sodium is sprinkled over the plates by squeezing through canvas, and this mercury is then rubbed in hard by means of the "rubber" already described, beginning at the top of the plate so as to distribute the pasty amalgam as evenly as possible all over the plates. When the plates have become "set," and it is desired to clean up amalgam from them, this is done before rubbing up. The tools used in cleaning up plates are "amalgam" knives (which are simply large-sized painters' palette knives, 6 to 8 inches long, made of good flexible steel, and kept scrupulously clean), scrapers made by turning over the end of a worn-out flat file for about half an inch and grinding to a chisel edge, and one or two sharp wood chisels about 1 inch to $1\frac{1}{2}$ inches broad. An enamelled iron cup forms the best recipient for the amalgam. Commencing at the bottom of each plate, two men, one on either side, rub all the amalgam upwards, using either stiff brushes, like scrubbing brushes, or the india-rubber "rubbers"; any amalgam that may adhere too firmly to be thus dislodged, may either be softened by a few drops of mercury, or loosened with the scraper or wood chisel; the whole of the amalgam on each plate is thus swept into a heap, and then transferred to the cup by means of the amalgam knife. Care must be taken not to scrape too closely, so that enough gold amalgam may be left to form a film all over the plate. When the clean up of

each plate is finished, a little mercury is sprinkled over the plates, and they are rubbed down as before. An objection to the use of stepped tables is that these take a little longer to clean up than do plain tables; but in my opinion this objection does not by any means outweigh the advantages of the former system. The amalgam is locked in a safe and reserved for after treatment. The whole process of cleaning and rubbing up a 15-foot plate should not occupy two men more than ten minutes, and rubbing up alone, five minutes, from the time of stopping the battery to that of restarting it. Usually this opportunity is also utilised for changing screens, guide blocks, &c., or making any small repairs, that may be needed, to the mill.

Gold amalgam gradually accumulates on the plates, which cannot be removed by the daily clean up; this is usually taken off once a month, when the plates are thoroughly cleaned. For this they are sometimes taken off the tables and gently heated so as to soften the amalgam, and then rapidly scraped. A recent American improvement on this practice consists in softening the amalgam by dipping the plate into boiling water. A still better plan in mills driven by steam power is to heat up the plate in place by turning a jet of steam on to it, covering it at the same time with a wooden hood to keep the heat in. The amalgam may also be softened by means of mercury, and the plates thoroughly scraped with wood chisels without removing them from the tables, but great care must be taken not to go too deep, so as to avoid scratching or cutting the plates. A film of gold amalgam will always adhere so intimately to the copper as not to be removable in the ordinary way, and this has to be taken off by "burning." The copper plate is taken off

the table and heated sharply over a fire of wood-chips, &c., till all the mercury is volatilised, care being taken not to inhale the fumes; the gold can then be stripped off in a more or less coherent film. After "burning," the plates must be re-silvered or amalgamated like new plates. They always, however, retain some gold, which appears to soak in by some form of molecular action until it becomes truly alloyed with the copper. Hence old mill plates have an intrinsic value considerably above that of the copper they contain, and in selling them this must be taken into account. It is perhaps best to have them melted down and to assay the resulting ingots. This circumstance that old mill plates contain valuable quantities of gold is one which is occasionally overlooked, and instances are on record where the purchasers of an old mill have realised very handsome profits on their bargain through such neglect on the part of the vendors.

Some interesting figures, which must, however, be looked upon as exceptional, have been recorded by Mr. R. T. Bayliss [1] at the Montana Mining Company's mill, Montana. A copper-plate, 8 feet by 4½ feet, plated with 1 ounce of silver to the square foot, was in use 3 years and 10 months, during which time 14,942 tons of ore were run over it. Outside amalgamation alone was used, and the plate was only rubbed up and never scraped, until at the end of this period a scale of hard amalgam 0·16 inch thick had accumulated upon it. By daily rubbing this plate had yielded 6,426 ounces of bullion assaying 541·5 of gold and 443·9 of silver per mil. It was then "burnt," when it yielded 866·1 ounces of bullion that assayed 431·4 of gold and 562·5 of silver

[1] *Trans. Amer. Inst. Min. Eng.*, vol. xxvi., 1897, p. 33.

per mil. After "burning," the copper-plate was melted up and was found to contain 8·96 ounces of gold, or in other words the plate had absorbed about $\frac{1}{4}$ ounce of gold to the square foot, which could not be removed by any mechanical treatment short of removing the material of the plate itself.

The amount of gold superficially retained by copper plates has also to be taken into account in custom milling; it is usual to allow the owners of the stone which is being milled to scrape the plates for themselves, but they are not allowed to heat them, and, of course, must not scrape down to the bare copper, but must always leave such a film of amalgam on the plates as to keep them in working order. This is a matter of little or no importance when large lots are milled, but in the milling of small test parcels of a few tons, the results are liable to be rendered utterly untrustworthy, because it is impossible to be sure that the amounts of gold amalgam at the commencement and end of the run shall be exactly the same. This very obvious source of error is frequently overlooked in test-crushings, which should never be made on a less quantity than 25 tons at the least, whilst the whole parcel should be put through one mortar box only.

Quantity of Water.—The amount of water required for a stamp-mill varies within very wide limits, depending upon the character and composition of the ore, the degree of fineness to which it has to be crushed, and the efficiency and arrangement of the battery. Average figures have already been given on page 263, and it can only be added that the exact amount required in any case can only be determined experimentally. As already pointed out, the chemical composition of the battery

water is no less important than its freedom from suspended clayey matter or slime; free acid, for example, should always be neutralised, and greasy water, *e.g.*, condenser water from an ordinary condensing engine, should on no account be used; if its use is absolutely unavoidable, it should be treated with lime, and the precipitate formed allowed to settle before it enters the mortars. An ore that contains much clay will require more water than a comparatively clean quartz.

In order to be able to regulate the water supply with great accuracy, the valves controlling the supply to each box should be fitted with hand-wheels, projecting over the front of the mortar-box; the mill man can then watch the effects on his plates whilst he is in the act of setting his valves. It is an advantage to have these hand-wheels roughly divided by marks, say at every ten degrees, so that they can be at once set to any desired position when restarting the mill after a stoppage.

Temperature of Water.—This is an important point in the management of copper-plates. A high temperature favours amalgamation, but at the same time tends to soften the amalgam and make it so fluid as to be readily carried off from the plates by the pulp. A low temperature, on the other hand, not only retards amalgamation, but also renders the amalgam so brittle that it is apt to crumble to powder, and thus be carried off and lost. The most suitable temperature is about 60° to 70° F. As a rule, it rarely rises above this, because the pipes bringing water to the mill are mostly (and should always be) covered up so as to be protected from the direct rays of the sun. In cold climates it is necessary to heat the battery water in winter time, and provision must be made for this when putting in the water mains.

Mercury Wells.—These are sometimes used for amalgamating the gold in the pulp; their general arrangement consists of a shallow trough filled with mercury, having in its centre a baffle board, the lower edge of which just dips into the latter; the pulp has thus to force its way down into intimate contact with the mercury before it can escape, and, during this contact, amalgamation of the gold in the pulp takes place, the amalgam so formed sinking to the bottom of the well. Mercury wells are much more largely used in Australia than in America, but they are gradually being dispensed with even there. When used, they are now practically always combined with amalgamated copper-plates. There are numerous and serious objections to wells. In the first place, they require a large amount of mercury, in which a certain amount of capital has to be locked up. Where so much mercury is in use the loss is bound to be proportionately great, and the mercury that is lost is sure to carry off some gold with it. Moreover, the well is not a very effective amalgamator, as the surface of mercury which it exposes to the pulp is small; it is doubtful whether any mercury well exposes as much surface of mercury to the pulp as a table 12 inches long would do.

One of the best forms of mercury well is shown in the annexed figure (Fig. 81). It is made by Messrs. Appleby Bros., Limited, of London. The material is cast-iron, and it is enamelled on the inside. The baffle board slides in a groove, and can be fixed by wedges at any desired point. Screw plugs are provided in the bottom by means of which the mercury and amalgam can be drawn off when it is desired to clean out the well. Mercury wells are frequently made of hard wood, and I have employed wells made of stout sheet copper

amalgamated on the inside, so as to enable the mercury to moisten the walls of the well, thus preventing the accumulation of sand, &c., on the bottom and in the corners of the well, which otherwise often takes place. Whatever the material of the well, the principle is exactly the same. The point that requires most care in the manipulation of the mercury well is the position of the baffle board; if this be too deep in the mercury, the pulp will be unable to balance the column of mercury so formed and the well will be sealed, with the result that no pulp at all will pass through it; if, on the other hand, the baffle board is not immersed sufficiently, the pulp will simply slide over the surface of the mercury without stirring up the latter at all, or coming into intimate contact with it. The correct position for

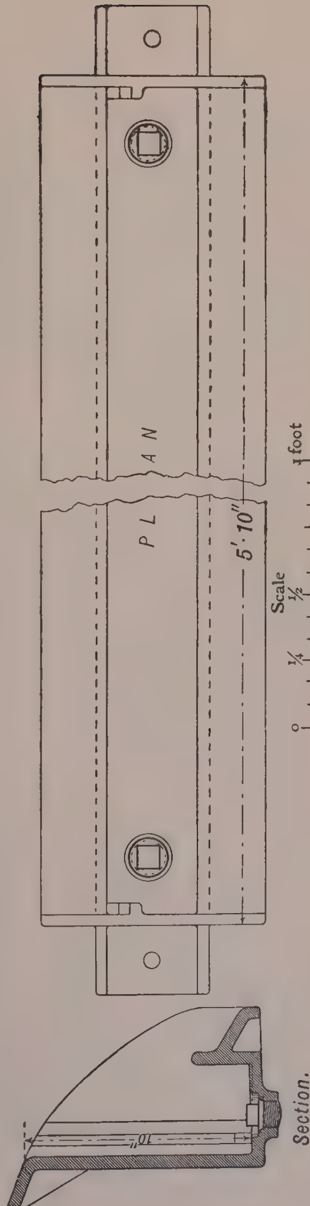


FIG. 81.

the baffle board can, however, be very easily determined, even theoretically.

Let Figs. 82 and 83 represent diagrammatically cross-sections of a mercury well before and during the passage of the pulp respectively, the mercury being distinguished by the shaded, and the pulp by the dotted portions of the diagram. Let the height of the inflow of pulp above its overflow be called a, the height of mercury above the bottom of the baffle board when the pulp is flowing b,

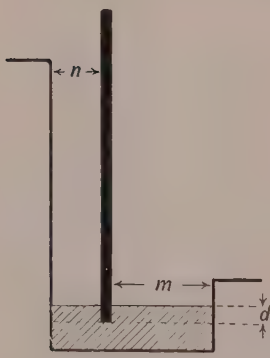


FIG. 82.

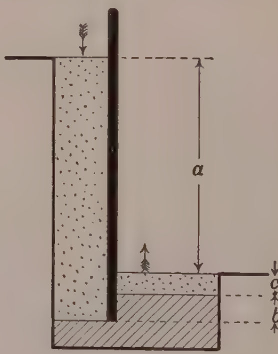


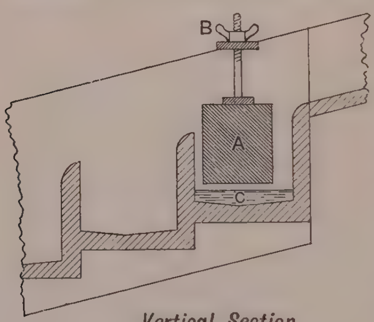
FIG. 83.

and before the pulp flows d, and the height of the escaping pulp above the level of the mercury c; also let n and m be the respective widths of the inflowing and outflowing sides of the well. Then in Fig. 83 it is necessary, in order that the pulp may flow, that the descending column should be heavier than the ascending one.

Therefore $a+b+c>c+13{\cdot}6b$ (the specific gravity of mercury being taken at 13·6) or $a>12{\cdot}6b$. In Fig. 82, if the baffle board be immersed in the mercury to a depth d, it is clear that the column of mercury whose

depth is d must be transferred from the inflowing to the outflowing side; when thus transferred its depth will be $\frac{n}{m}d$, so that the total difference in height between the two columns of mercury will be $\left(1+\frac{n}{m}\right) d$. Therefore $b=d\left(1+\frac{n}{m}\right)$. As the pulp will only flow whenever $a>12{\cdot}6\left(1+\frac{n}{m}\right) d$, the maximum possible value of d is given by the expression $\frac{am}{12{\cdot}6(m+n)}$. This expression accordingly indicates the greatest depth to which the baffle board may be immersed in the mercury; it is not strictly accurate because various factors have been neglected, such as the capillarity of the mercury, the weight of sand carried by the water, the thickness of the baffle board, and the depth of the layer of pulp passing beneath it. These errors, however, partly balance each other, and the above formula may be used as a general guide. To take a concrete example: with the well shown in Fig. 82, $a=9$ inches, $n=2$ inches $m=3{\cdot}5$ inches, and therefore the greatest possible value of d will be $\frac{9}{12{\cdot}6\left(1+\frac{4}{7}\right)}=0{\cdot}45$ in. A convenient modification of the mercury well, much used in Hungary, is shown in section in Fig. 84. In this form the well is wider and deeper than usual, and the baffle board is replaced by a rectangular block of wood (A), the position of which is regulated by a couple of thumb-screws (B). This arrangement forms a U-shaped channel through which the pulp has to travel, the bottom of the horizon-

tal portion of the U consisting of a surface of mercury. There is a depth of $\frac{1}{2}$ inch of mercury (c) in this well, which is constructed of sound, hard wood, and the block is screwed down to within a $\frac{1}{4}$ inch of it. The width of the channel on the inflowing side is $\frac{3}{4}$ inch, and on the outflowing side $\frac{3}{8}$ inch. Thus arranged, this well has a capacity of $1\frac{1}{2}$ cubic feet of pulp per minute for each foot in length of the well, and this can be increased, if necessary, by lifting the block by means of the thumb-screws. A series of these wells is usually arranged one below the other as shown in the section. The mercury and amalgam can be drawn off when desired through a pipe at one end of the mercury well, the bottom of the well sloping gently in all directions towards this pipe. Wooden mercury wells are always best cut out of the solid.



Vertical Section

Scale 1½" to 1 ft.

FIG. 84.

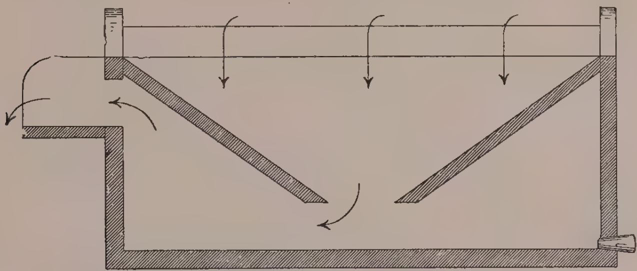
When in operation, amalgam gradually collects in the wells; this sinks to the bottom unless the ore be very rich in silver. The gold-silver alloy containing two parts of silver to one of gold has just about the same specific gravity as mercury, so that all alloys containing less gold than this proportion will float on the surface of the well instead of sinking. Such an alloy, however, is but rarely found occurring in gold ores. When much amalgam has accumulated in the well, it is best removed.

This may be done by means of a small scoop pierced with holes, but is usually done by hand, the amalgam being roughly squeezed, and the excess of fluid mercury returned to the well. If necessary, fresh mercury is then poured in to replace that so removed. The surface of the mercury in the well is apt to become foul and dirty. When this occurs the mill must be stopped, and all sand, &c., washed as completely as possible off the surface of the mercury, which is then skimmed by means of a piece of indiarubber, or of thick blanketing, until the surface is once again quite bright. The skimmings thus obtained are put on one side for subsequent treatment. When a complete clean-up is made, all the mercury and amalgam are removed from the well by means of the screw-plug, or else by means of a small scoop if the well is not furnished with a plug, the entire contents being collected in mercury pails. These are best made of stout enamelled iron, and should be fitted with a strong iron strap going right under them from side to side, and furnished at the upper end with eyes in which the bale works; this is better than having the eyes merely riveted to the pail, as much strain is thrown on them. About 1 to 1½ gallons is a suitable size. It should be remembered that if the mercury is allowed to run into a bucket half filled with water, there will be no loss of mercury by splashing. The contents of the mercury well are taken into the mercury room, skimmed and allowed to stand for about a day. At the end of this time the fluid mercury may be poured or syphoned off from the semi-fluid amalgam which will have settled in the bottom of the pail. This fluid mercury must be purified from base metals as previously recommended, charged with a little sodium, and may then be used over again. It will still

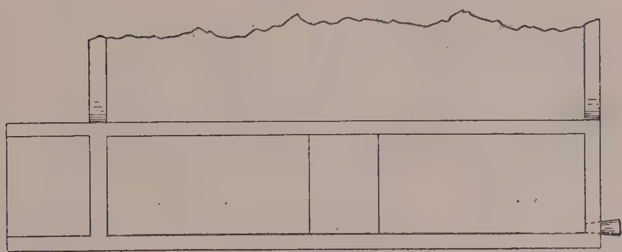
be fully saturated with gold amalgam, but this is an advantage in working the wells. These cleans-up must be made once a week to once a month, according to circumstances. It is more difficult to guard against the theft of amalgam when wells are used than in the case of tables, as the former give no indications whether they have been tampered with or not.

Mercury Traps.—It is usual, and very advisable, to place at the foot of the last copper-plate an arrangement for catching any stray particles of mercury that may have escaped with the pulp. The usual form consists of a box into which the pulp is allowed to flow, the construction being such that it must go down to the bottom of the box and then rise again before it can escape. The heavier particles, and amongst them mercury, will collect in the bottom of the box, the lighter portions of the pulp flowing away. The contents of the box are emptied out from time to time, and put aside for special treatment. Mercury traps may be made either of wood or of iron; they should be of fairly large sectional area so that the rising column of pulp shall not move too rapidly and have time to deposit all the heavier particles. A wooden and a cast-iron mercury trap are shown respectively in Figs. 85 and 86; their construction is so simple as to need no explanation. Sometimes two mercury traps, one below the other, are employed. When the pulp leaves the mercury trap, the whole of the free amalgamable gold is supposed to have been extracted from it, and the process of extracting the combined gold then commences. A certain portion of the mercury still escapes with the pulp, and some, but not much, is caught in the process of concentration. Some is, however, so completely floured that the minute particles,

to which it is reduced, defy all attempts at collecting it. The value of the mercury thus lost is not very great, but its loss always entails some loss of gold, as the escaping mercury is probably fully saturated with the precious metal. It is, therefore, necessary to keep



Vertical Section.



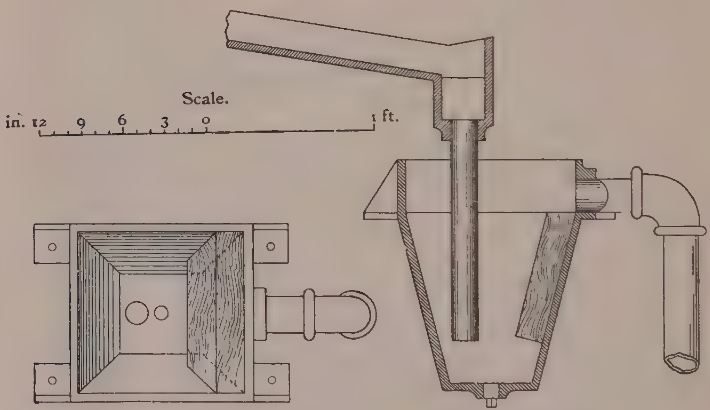
Plan.

Scale, 1″ = 1′

FIG. 85.

this loss within the narrowest possible limits. Its amount varies very greatly, depending principally upon the methods of amalgamation employed. It is probably least when outside plate amalgamation alone is practised, and greatest when mercury wells are used. Recorded

losses vary between 0·001 lb. and 1·0 lb. of mercury per ton of ore crushed. The losses in the El Callao mill, which may be looked upon as a type of a well run mill, were about 0·02 lb. per ton of ore over an entire year, and in the Treadwell-Alaska mill 0·01 lb. The average loss when inside amalgamation and copper tables are employed may be put down as 0·03 lb. per ton, or say, a pound of mercury for every thirty tons of ore milled.



Plan. *Vertical Section.*

FIG. 86.

This loss of mercury represents a cost of about $\frac{2}{3}d.$ per ton; in some special cases it has risen to as much as $6\frac{1}{2}d.$ per ton, but this figure is altogether exceptional, and indicates very faulty work. In California 1 lb. of mercury to fifty tons milled seems to be about the average; in Victoria some of the best run mills appear to lose 1 lb. to thirty-five tons, whilst in New Zealand the loss seems to be twice as high. In order to keep a check

on the loss of mercury, a special mercury book should be kept in which the weight of mercury supplied from time to time to the mill during the run shall be entered, together with the date, and debited to the mill. When a complete clean-up is made, all the mercury collected from all sources must again be weighed and entered, on the credit side; the difference will of course be the loss during that run. It will thus be seen whether the loss is either excessive or irregular, and in either case it should be promptly investigated, so as to guard against the loss of gold that may accompany it. A mercury book systematically kept affords, moreover, one of the best checks on any thefts of amalgam that may be taking place in the mill

CHAPTER XI

PRINCIPLES OF CONCENTRATION—RIFFLES—BLANKETS—BUDDLES—SHAKING TABLES—VANNERS—SIZING—SPITZKASTEN

Further Treatment of the Pulp.—When the pulp escapes from the amalgam traps it is supposed to have all its free gold completely removed from it; if the ore originally contained only free gold, or so little combined gold as not to be worth saving, the pulp runs directly from the traps into a launder, which carries it to waste in some convenient spot. It is needless to point out that in the selection of a mill site, ample provision should be made for a tailings dump leading into some ravine, swamp, or river, which is not liable to be choked up. Should the pulp, however, contain combined gold (using the word "combined" merely as a convenient expression to signify gold that refuses to amalgamate by simple contact with mercury, and not as necessarily indicating that it is in a state of chemical combination), then the pulp will have to undergo a further process in order to recover this combined gold. This object can be attained in two ways: firstly, by means of concentration, which collects the auriferous portions of the tailings into a comparatively small bulk for further treatment; or, secondly, by means of the more modern method of

treating the tailings before or after such removal of concentrates by a chemical process, as, for instance, by cyanide extraction.

Concentration.—A study of Table C, page 10, which includes all those minerals which carry gold with them, will show that all these are comparatively heavy ones. The specific gravity of gold-bearing minerals may be taken at 5·5, or a little higher; whilst that of the non-metallic and worthless portions of the pulp, which will consist of some or all of the minerals in Table A, may be taken at about 3; owing to their high specific gravity, some of the minerals enumerated in Table B will usually accompany the gold-bearing minerals. The object of concentration is, accordingly, to separate all the minerals of specific gravity of 5 or over from those of 3 or under.

The mechanical principles upon which all concentration is based may be very briefly and simply stated. All bodies that are acted upon by any force would be propelled by this force at equal velocities, if friction did not exist, and if there were no resistance to their movement. This theoretical condition is, of course, never realised, as every body meets with more or less resistance from the medium which surrounds it; its motion being impossible except by displacing the particles of this medium, more or less of the force causing motion is absorbed in performing this latter operation, the amount absorbed being greater, other things being equal, the greater the surface of the body in question. Accordingly, when a body is immersed in a fluid, whether gaseous or liquid, the particles of which offer resistance to its free motion, then the velocity imparted to it by a given force varies with its mass and inversely with its surface. Assuming the bodies to be spherical, the mass will be a

function of d^3s, where d is the diameter of the sphere and s the specific gravity of the substance composing it, while its surface varies inversely as d^2; hence the velocity imparted to the body will be a function of ds. In order that ds shall remain a constant, if s be diminished d must be proportionally increased, and *vice versa*; in other words, a given force will propel a smaller body of high specific gravity with the same velocity as a proportionately larger body of lower specific gravity, when both are suspended in the same fluid medium, and, under these circumstances, a heavy particle will move faster, or will move through a greater space in the same period of time, than will a light particle of the same size. Therefore, when a number of particles, approximately spherical and approximately uniform in size, are subjected to the action of a force, which may be gravity or may be some mechanical impulse, which shall tend to move them in a given direction, the heavier particles, moving further in the same space of time, can thus be separated from the lighter particles. The above statements are true of particles approximately spherical, but it has already been pointed out (page 312) that the shape of a particle has a most important influence on its behaviour when suspended in a liquid; so that sufficiently thin sheets, for example, have no tendency to sink in a fluid, however much greater their specific gravity may be than that of the suspending medium, on account of the very high ratio which their surface bears to their mass; whilst in spheres, where the ratio of surface to mass is a minimum, the relative effect of varying specific gravity is most readily apparent. Successful concentration, therefore, can only be effected when the particles of crushed matter to be treated approxi-

mate to the sphere in shape; the object of a crushing machine for preparing pulp for concentration should therefore be to granulate the material as far as possible. In the stamp-mill this object is never very well attained, but the best results are produced when the crushed material is discharged with the greatest rapidity from the battery box. Inside amalgamation, when inside copper-plates are used, is accordingly prejudicial to good concentration, as it precludes working with a low discharge and a narrow mortar box, which are, as we have seen, the conditions which promote rapid discharge of the pulp.

Concentrators.—The number of appliances designed for concentrating tailings from stamp-mills is something enormous, and new modifications of them are continually being brought out. The number of those that have successfully withstood the test of experience, and have held their ground for any length of time, is by no means great; all are reducible to a few simple types, which may be classified as under:—

1. Appliances in which the heavier particles are simply allowed to settle under the action of gravity; to this class belong strakes, riffles, blanket-tables, &c.

2. Appliances in which the action of gravity is assisted by external means, as in the case of buddles.

3. Appliances in which an impulse is communicated to the particles by mechanical action, as in the case of various forms of shaking tables, rotating concentrators, vanners, &c.

Class I.—Perhaps the simplest, though at the same time the least efficient, method of collecting concentrates, or sulphurets as they are often called, consists in employing riffles set in the launder through which the taillings

run to waste. By this means only the coarser portion of the heavy minerals is collected, and hence this system is never used when the sulphurets are of any considerable value, although it has its uses in helping to catch a certain amount of these as well as of globules of mercury and particles of amalgam that may have escaped the trap. The simplest form of riffle consists of slates of wood about 1 inch wide and 1 inch to 1½ inches deep, pinned or wedged firmly in the bottom of the launder at distances of from 6 inches to 2 feet apart, according to the amount of substance it is desired to catch. The sands are caught temporarily against each of these riffles, and being kept in constant agitation by the force of the stream of pulp, the lighter particles remaining longer in suspension are washed over the top of the riffle by the current, whilst the heavier and coarser particles falling more rapidly to the bottom of the stream, accumulate against the riffle. From time to time the riffle-launder is cleaned out, usually when the mill is stopped for cleaning up the plates. This is usually done by placing a bucket at the end of the riffle-launder, lifting the riffles, and washing the accumulated material into the bucket by means of a hose pipe carrying a small stream of clear water. A launder about 2 feet wide and 6 inches deep is usually ample for the pulp from ten heads of stamps; it is generally set on a steeper grade than the copper tables, say between 2 and 3 inches to the foot. The wooden riffles are sometimes protected from wear by a strip of thin bar iron screwed to their upper surface. Sometimes short lengths of very light rails are used instead of wooden riffle bars, or else wooden slats like venetian blinds set in frames (so called Hungarian riffles), or grid-shaped cast-iron riffles are used. Round riffles consist of holes abou 1½ to

2 inches in diameter bored through short pieces of $1\frac{1}{2}$-inch plank, which pieces of plank are then laid in the bottom of the launder. The holes are usually arranged quincuncially, the pieces of plank being about 3 or 4 feet long. A space of an inch or two is mostly left between successive planks to act as a transverse riffle. At times longitudinal riffles are used, but these are not very effective and cannot be recommended for this work. The sands so collected are usually piled up and kept for further treatment; they are often subjected to further concentration in buddles, just as blanketings are in order to obtain cleaner products.

Blanket Strakes.—This primitive, but nevertheless efficient method of obtaining concentrates, is still largely used. It consists of a shallow launder, the bottom of which is covered with a series of pieces of blanket or some similar material. When the stream of pulp is sent over these, the heavier particles, falling more rapidly than the lighter ones, settle on the blanket and are entangled in the nap, being thus held whilst the lighter particles are carried off in the current, which is, of course, strongest and most rapid at the surface of the stream, and slowest at the bottom. As soon as the nap of the blanket has become filled with particles of mineral, it presents a smooth surface, and is naturally unable to catch any more; before this stage is reached, it is accordingly necessary to wash out the particles of mineral from the blanket, and to expose a clean blanket surface to the stream of pulp. In order that the process may continue uninterruptedly, it is necessary to have duplicate blanket strakes, one of which may be filling, whilst the other is being washed. Usually the pulp from each five-head battery (or sometimes from each ten

heads) is run over a set of strakes, consisting of two launders side by side, or of one wide launder with a division down the middle, which thus makes it equivalent to two launders. Their length is usually 12 to 20 feet, and their width 24 to 36 inches, 30 inches being often adopted. Each blanket should be 4 to 6 feet long, there being three or four in each strake. These must be laid so as to overlap at least 6 inches, the lower end of the upper one, of course, overlapping the upper end of that next below it, like tiles on a roof. At the head of each strake there should be a head-board with pins, so as to distribute the pulp quite uniformly across the width of the strake, this being a very important matter to attend to. Each strake is also supplied with a gate by which the pulp can be shut off from it. When the pulp leaves the amalgam traps, it usually runs into a transverse distributing launder, which should be supplied with water-pipes fitted with valves, so that clear water may be run into it, in order to dilute the pulp to any desired extent. This is usually necessary, for a consistency of pulp, which works well on the copper-table, is usually too thick for a successful concentration on blanket strakes. It will generally be found to require about half as much clear water, and sometimes as much again as it already contains. It is important that the tailings should flow in a thin, fairly rapid stream, and too much care cannot be taken to secure its uniformity across the width of the strake. Blanket strakes should be carried like the copper-tables on folding wedges, or else on screws supported by stout trestles well sunk in the ground, and should be adjusted so as to be accurately horizontal transversely, whilst the grade can be varied at will. A good plan is to have the upper end of the strakes supported on some form of hinge,

whilst the lower end is capable of adjustment as above. The grade is generally flatter than that of the copper-tables, being mostly between $\frac{3}{4}$ inch and 2 inches to the foot. No definite rules can be laid down for it, as it depends on many circumstances, such as the percentage of sulphurets present in the ore, their specific gravity, the fineness of the particles of crushed ore, and the general character of the latter. It must be determined by experiment in each individual case what grade of strakes and what degree of dilution of the pulp gives the cleanest concentrates, together with a minimum of loss of sulphurets. The result to be aimed at depends upon the value of the concentrates, and the after treatment which these are to undergo. Thus if these are very rich and are to be treated on the spot, say by chlorination, it will probably prove economical to aim at saving the largest proportion possible, although by doing so a less clean product may be the result; that is to say, a certain proportion of valueless quartzose sand may still remain mixed with the concentrated sulphurets. On the other hand, if the sulphurets are comparatively poor and have to be shipped for treatment, it may be advisable to aim at obtaining as clean a product as possible, at the expense of losing a certain proportion of the sulphurets in the escaping tailings. These results depend not only upon the conditions already referred to, but also upon the frequency with which blankets are washed, frequent changes favouring the former, and long intervals the latter form of product. Blankets have to be washed at intervals varying from one to four hours according to circumstances, in a trough provided for the purpose. There should be one trough to every two sets of strakes, and they should be set opposite their lower ends. These troughs are made of plank

usually about $1\frac{1}{2}$ inches thick, well bolted together, a strip of tarred blanket being interposed between the edges, so as to make them perfectly water-tight. This is a better plan than employing tongued and grooved joints, as in case of leakage the former system admits of caulking, whilst the latter does not. The trough is furnished with one or two shelves about 18 inches wide, running along the sides, and so placed as to drain into the trough. The general arrangement is shown in Fig. 87.

To work the blanket strakes, the blankets are first of all thoroughly soaked in water, and then laid in place, in one (say the right-hand one) of each set of strakes, arranged as already indicated. They must be thoroughly wet through, or else they are apt to shift when the stream of pulp strikes them. The pulp is then admitted, great care being paid to the evenness of its distribution, so that no channels nor local accumulations of sand are ever allowed to form. At the expiration of the proper interval of time —when the top blanket has taken up all the sulphurets it can hold—the gate leading to the other (the left-hand) strake of each set is opened, and the right-hand gate shut down. Each blanket is then folded, beginning with the top one ; the lower end of each blanket is lifted and folded over by bringing it to the upper end, and the lower end, thus doubled over, is again brought to the top, and so on till the blanket has been folded four or six times, making a convenient bundle to carry, and allowing none of the concentrates to escape. The second blanket is similarly folded, and so on ; often the lower two blankets (when there are four in each strake) only need washing half as often as the upper two ; in that case the lower blankets need not be disturbed. When all the blankets are folded, they are carried to their respective troughs and laid on

one end of the shelf; they are then thrown one by one into the trough (which must be nearly full of water), unfolded, and washed by being moved rapidly to and fro, the side which has been uppermost in the strake being lowermost in the trough. When all the mineral has been washed out, the blanket is folded again and laid on the other end of the shelf until all are washed; they are then replaced in the strakes ready for use again as soon as the left-hand blankets require washing; and so the process continues. The blanket trough should be large enough to contain a week's accumulation of blanketings

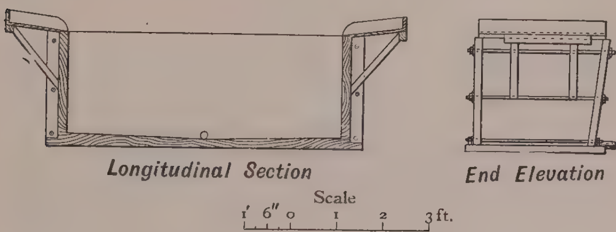


FIG. 87.

without being more than half full. The troughs should be emptied every week, and the blanketings stacked for further treatment. Care should be exercised in the selection of the blankets; several firms manufacture special qualities of blanketing for this work, which can be obtained from any makers of mining machinery. The blanket should be not very coarse, strong, and closely woven, with a short, stiff nap; a long nap is apt to become felted very quickly by the running pulp, and will hold a very small quantity of sulphurets. It is false economy to buy cheap blankets of inferior quality, as they wear out very rapidly; in cases of emergency, however, any

blankets may be used, such, for instance, as the trade blankets that are sold for the use of natives in most semi-civilised countries. Gunny sacking is sometimes used instead of blankets, and is recommended by some authorities when the sulphurets are very finely divided. Many other substitutes have been employed, notably plush, which, although very expensive, is said to answer extremely well. A good quality of blanket may be bought from about 1*s*. 3*d*. to 1*s*. 6*d*. per yard. The length of time that a set of blankets will last depends upon so many circumstances that it is scarcely possible to give even approximate data; their life, however, rarely exceeds one month. The washing of the blankets is usually done by boys; each boy can easily attend to three sets of strakes unless the proportion of sulphurets is unusually high. These two items—the cost of labour and the wear and tear of the blankets—are the principal ones in the cost of blanket concentration, and these will vary so widely in different localities that no price generally applicable can be fixed. Concentration by blankets used to be far more generally employed in gold mills than it is now, having been to a great extent replaced by more perfect mechanical methods; the great advantage it presents is that the first cost of the installation is extremely low as compared with concentrating machinery, whilst, on the other hand, it is comparatively expensive, though imperfect in its operation. It can only be recommended, therefore, in the case of small mills where there is not sufficient capital to equip them with better appliances, and in new mining districts where the value of the concentrates requires to be proved before any large sums of money are invested in concentrating machinery.

Canvas Tables.—In Australia blanket strakes are

often followed by canvas tables; these are long broad strakes 80 to 100 feet long and about 4 feet 6 inches wide set at a grade of $\frac{3}{4}$ to $\frac{1}{8}$ inch to the foot. In these are laid light wooden frames to which canvas is nailed; finely divided pyrites settles readily upon and adheres comparatively firmly to the surface of the canvas, and when a set of frames is charged with pyritic slimes, it is removed and the pyrites washed off. Three tables are usually arranged side by side, two of which are taking the pulp whilst the third is being cleaned. Each table of the above dimensions will carry about 30 cubic feet of pulp per minute, and one set of tables is considered sufficient for a mill crushing 100 tons per day. It is found that canvas tables will catch pyritic slimes that escape from ordinary blanket strakes; they are much in favour with the Chinese in Australia, who at times arrange to be allowed to treat by these means the tailings escaping from stamp-mills; they are hence, at times, spoken of as Chinese tables.

In California very similar canvas plants are employed, following usually the vanners or other concentrators; here, however, a large number (up to 90) of narrow strakes 12 to 22 inches wide are preferred; the tables are from 20 to 100 feet in length, and their grade varies from $\frac{3}{8}$ to $\frac{3}{4}$ inch to the foot; they are laid with light canvas in the same way that blankets are arranged in blanket strakes. In Tuolumne County one canvas plant has an area of 2,400 square feet of canvas tables to treat 25 tons of quartz crushed in 24 hours, the stone carrying 3 per cent of pyrites. The first cost of these plants is very small, and the chief expense in running them is the labour required, as the canvas lasts a long time.

It would seem as though such canvas tables might with

advantage be replaced by Brunton canvas belts, which work automatically; as far as I know, however, the Brunton belt has never been used in connection with gold mining, though apparently well adapted to this purpose.

Treatment of Blanketings.—As a rule, the blanketings, obtained as above, contain a very large proportion—as much in some cases as 75 per cent., or more—of worthless sand, and it is advisable to remove the greater part of this, so as to obtain comparatively clean concentrates, as a preliminary to further treatment. This is conveniently done in an ordinary square buddle. This is shown in Fig. 88; it consists of a rectangular box 8 to 12 feet long, 3 to 5 feet wide, and 1 foot 6 inches to 2 feet 6 inches deep. This box is set on a steep grade, the fall being between 1 and 2 feet in the length of the buddle. At the upper end there is a head-board about 15 inches long, extending the full width of the buddle, which is fitted with movable pins in order to regulate the stream of pulp. Above the head-board is a box into which the blanketings are charged with the shovel, a stream of water being also supplied so as to form a pulp of suitable consistency. The tail-board of the buddle is pierced with a double row of holes fitted with plugs so arranged that the water can be drawn off at any desired level. Across the buddle there is laid a plank on which the man working it stands; he is provided with a long-handled besom. Two men are required to each buddle; one charges the blanketings into the head-box, regulates the water supply, and attends to the pins on the head-board; the other handles the besom. The pulp thus formed on the head-board runs smoothly and evenly into the buddle, where the heavy constituents settle nearest to the head, the others being deposited towards the lower

end. By means of the besom the surface of the accumulating material is kept quite smooth and level, and no channels are allowed to form in it, the besom being worked from below upwards with a lateral movement

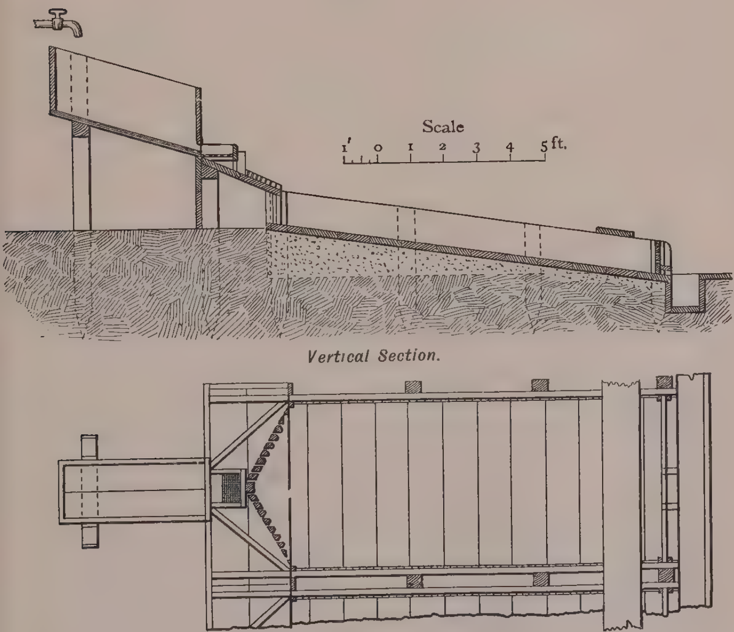


FIG. 88.

across the full width of the buddle. The success of buddling depends entirely upon keeping a smooth, uniform surface, and supplying the pulp in a thin, regular stream. As the surface of the sands in the buddle rises, the lower holes are successively plugged,

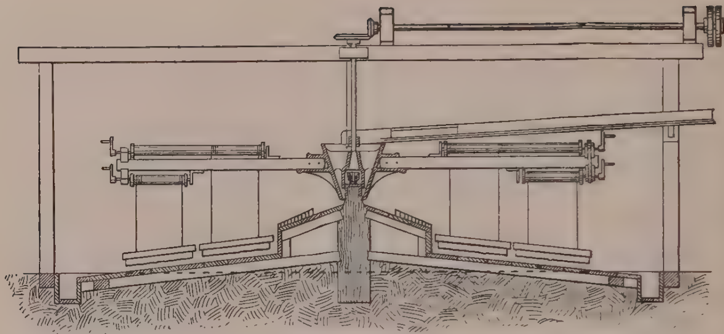
so as to keep the surface of the sands approximately parallel to the buddle bottom.

The water consumption of such a buddle is about 1 cubic foot per minute, which quantity of water should carry in 30 to 40 lbs. of blanketings. The buddle is filled to a depth of from 9 to 12 inches. When it is sufficiently filled, the water supply is stopped, and the bottom plugs taken out so as to drain off all the water. After this has been done, the contents of the buddle are divided into three parts—heads, middlings, and tails. The former, about a quarter of the whole, are fairly clean concentrates; the middlings, constituting about one-half of the total quantity operated on, have to be buddled again; and the tails are worthless and may be thrown away. This work is best done with long-handled shovels. Two men, working as above, can finish about four buddles in a ten-hours shift, putting through about 30 cwt. of blanketings at each operation. Instead of the square buddle, the round buddle or else the tye, to be presently described, may be used for the same purpose; the square buddle seems to give, however, the best results. If desired, the heads may be still further cleaned, by tossing or chimming, as in the Cornish method of tin ore dressing. To do this, the heads are charged into a strongly-made tub about 3 feet 6 inches in diameter two-thirds full of water. The pulp is stirred round vigorously with a long-handled shovel, and when in steady motion the shovel is withdrawn and the side of the tub struck steadily and rapidly with an iron bar about $1\frac{1}{2}$ inches in diameter, one end of which is resting on the ground. This treatment settles the sands, the heavier portions settling first, whilst the upper layers consist of worthless tailings; next comes a portion which requires re-treatment, and the clean

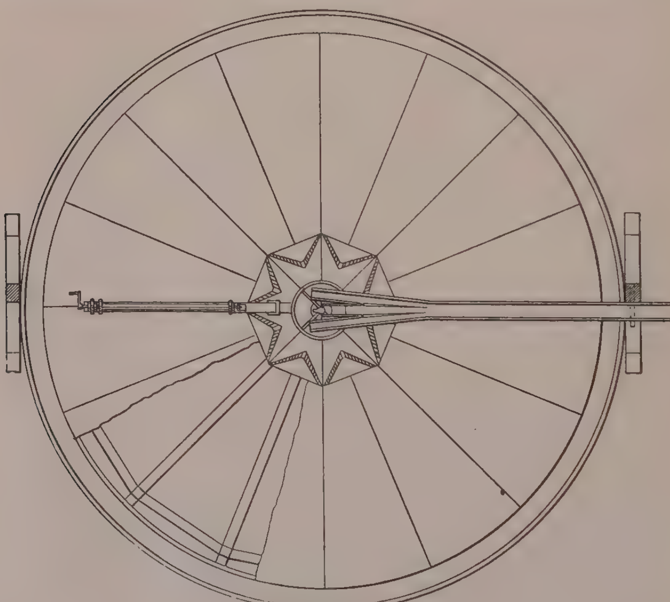
concentrates form the lowest layers. A tossing-tub, in which the stirring of the sands as well as the tapping to settle them, is done by mechanical means, has been successfully employed in some gold mills. It should be remembered that blanketings frequently contain a small proportion of floured mercury, which tends to run together into globules under the operations of buddling and tossing, and due care must be taken that these are not lost; they will, of course, be found in the very lowest layers of the cleaned concentrates, and can be collected by panning these.

Class II.—The square buddle above described can also be used to concentrate the valuable portions of the tailings as they leave the mercury traps. In this case no head-box is required, the tailing launders delivering direct into the head-boards of the buddles. They must be worked in pairs, so that one buddle may be filling whilst its fellow is being cleaned out. On this system two men are required to attend to each pair of buddles. Usually a third buddle is set close below each pair for the further concentration of the heads obtained; it is then convenient to make only two divisions of the buddle contents—namely, heads for re-treatment, and tails which are not worth the expense of further handling and can be thrown away.

For this class of work the round buddle is preferable to the square, as it does not require constant manual labour, but only supervision, so that one man can attend to several sets. These also must be built in pairs. The construction of the usual type of round buddle is shown in Fig. 89, which represents the ordinary convex buddle. Concave buddles are also sometimes, but more rarely, used, and there are numerous modifications of detail in



Vertical Section.



Plan.

Scale

0 1 2 3 4 5 6 7 8 ft

FIG. 89.

both types. The one shown here is, however, simple to construct and to operate, and fairly efficient. Its advantage is that it can be made by any ordinary carpenter and is inexpensive. It consists of an annular table in the shape of a very obtuse cone, the angle of which to the horizontal is between 4° and 7°. The outer diameter is between 18 and 24 feet, the inner diameter about 6 feet. The inner portion is occupied by a circular head-board, set at a steeper pitch than the buddle itself, and about 1 foot higher at its lower circumference. In the middle of this head-board is a conical hopper generally made of iron, which is nearly closed by a conical block so set as to leave only an annular space of 1 to 2 inches in width for the discharge of the pulp. This central conical block carries a vertical spindle, which is caused to revolve by bevel gearing, and to this spindle are attached two or four arms from which hang thin boards, the lower edges of which (set parallel to the face of the circular table) are furnished with brushes or strips of stout canvas or blanketing, which, like the besom of the workman in the square buddle, level off the accumulating sands to a smooth even surface. As the buddle gradually fills up with deposited tailings these brushes must be raised, which may be done automatically, but is usually performed by hand by means of some such simple device as is shown in the figure. It is easier to adjust these brushes if each is made, as shown, in two parts. The course of operation is precisely the same as that of the square buddle, and similar precautions have to be adopted to secure good results. The rate of revolution of the spindle is mostly from 10 to 12 per minute. A buddle such as is here shown can take 2 to 3 cubic feet of pulp per minute, whilst the degree of dilution of the latter should be such that each cubic foot

carries about 56 pounds of sand. The filling of the buddle takes from 2 to 3 hours, and it will then carry about 6 tons of concentrates; the weight of ordinary buddle concentrates or blanketings may be taken at about 100 lbs. per cubic foot. A pair of round buddles will mostly suffice for 10 heads of stamps. When the buddle is filled its contents are, as in the case of square buddles, usually divided into two—namely, the worthless tails, which are deposited at the outer edge of the table; and the heads, which compose the upper portion, and which require to be rebuddled, preferably in rectangular buddles.

Tyes are sometimes used for the concentration of low-grade sulphurets; these may be looked upon as a modification of the rectangular buddle. They consist of a rather flat-lying launder about 18 inches wide, 12 to 20 feet long, and 1 foot deep. The lower end can be closed by a series of narrow strips of wood about 1 inch deep, so that the out-flow can be raised to any desired height. The tailings are run into the launder, and a workman continually pushes the sand that accumulates in it, from the lower end towards the upper end with a light hoe, thus facilitating the carrying away of the lighter sand whilst the heavier particles accumulate on the bottom of the tye, the level of the outflow being gradually raised as the sulphurets accumulate in the tye. When the tye is full, the concentrates so obtained are worked over again, either in a similar but smaller tye with clean water or in a rectangular buddle. As in the corresponding case of buddles, these tyes must be worked in pairs. They are but little used, and cannot be recommended except for their cheapness.

Class III.—Nearly all the modern concentrating machinery used in gold mills belongs to this class, which

includes the various forms of shaking-table, of which there are endless modifications. Generally speaking, the shaking-table consists of a suspended inclined table upon which the tailings are delivered in a very thin stream. This table receives a rapid oscillating motion, the effect of which is to propel the lighter particles along one path, and the heavier ones along another, so that each may flow off into receptacles provided for the purpose. These machines are very effective and produce clean concentrates, whilst the loss in the tailings can be kept within very low limits. The original form of shaking-table is that devised by Rittinger, which has not been much improved on since, in which the direction of the shake is at right angles to the flow of the pulp. These tables are usually built in pairs, their general construction being shown in Figs. 90 to 92, which represent Rittinger's original design, the first figure giving a section, the next a plan, and the third a front elevation of the machine. The table is strongly made of hard wood, carefully planed and made as smooth as possible. It is suspended by four rods which allow its grade to be altered at will. The framing of the table is very substantial, and it carries a stout transverse piece near the middle, which receives a series of thrusts from a cam which pushes it against a strong wooden spring. The action of this spring throws it back sharply against a bumping-block, which gives the shake to the table, the shake thus consisting of a series of sharp shocks. Each table is usually 8 feet long by 4 feet wide, and has an inclination of from 3° to 6° to the horizontal. The average number of blows is about 100 per minute, but in working very fine sands as many as 150 per minute are sometimes given, the length of each stroke being about 1½ inches.

Above the upper end of the table is fixed a head-board with the usual distributing-pins. The pulp to be concentrated is delivered to one-fourth only of the head-board, the remainder being supplied with clear water. If the table were at rest, all the particles of the pulp

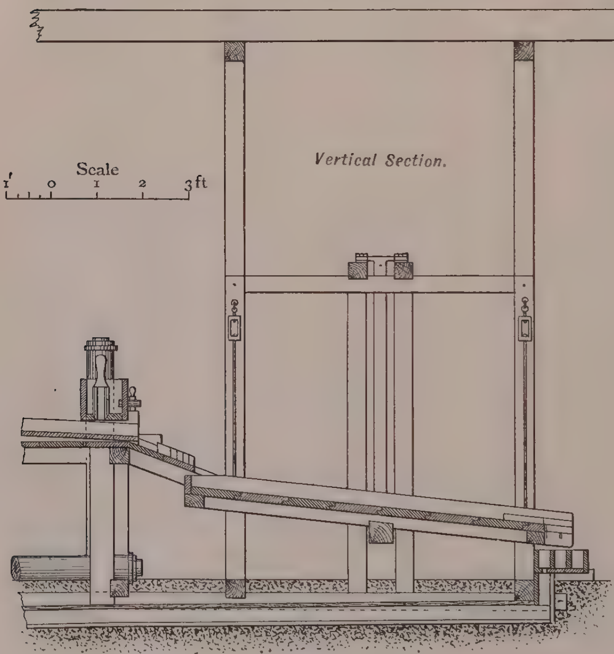
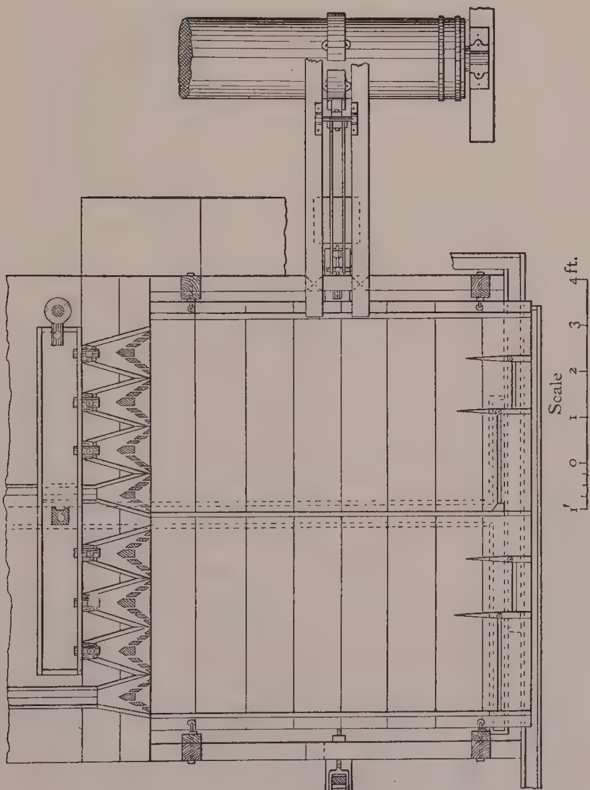


FIG. 90.

would tend to roll down the table in lines parallel to its length, the lighter particles achieving the journey more rapidly than the heavier ones. The impulse due to the shaking action acts in a direction at right angles to the line of flow, and hence the particles move down in a line,

which is the resultant of these two motions. As the velocity down the table is continually increasing, whilst the velocity in the direction of the shake remains constant,

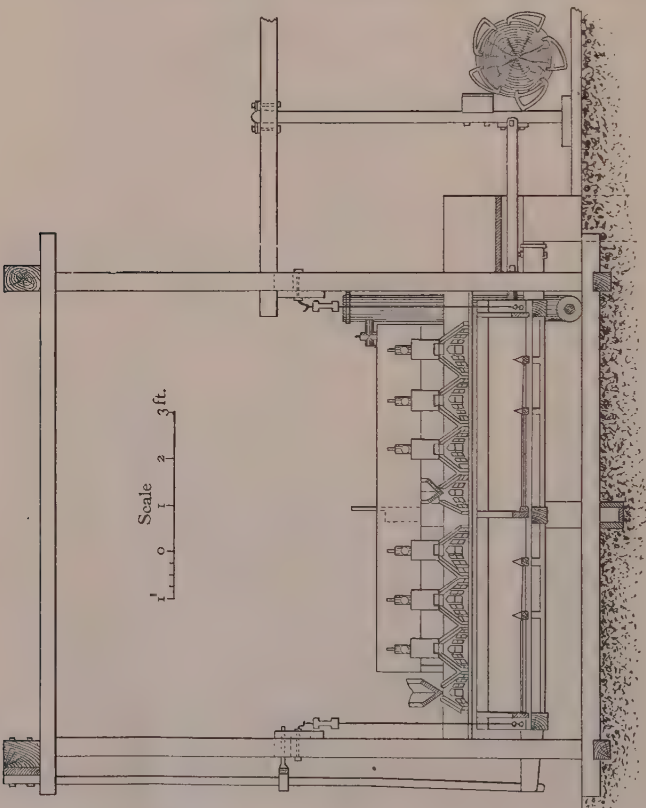


Plan.

FIG. 91.

this resultant takes the form of a parabola. The heavier particles moving more slowly, are exposed for a longer time to the action of the transverse impulses than are the

lighter ones, and hence are thrown further from the straight line of flow. It is thus possible to obtain a very complete separation of the pulp into concentrates and barren



Front Elevation.

FIG. 92.

sands, the action being quite continuous. This machine answers well, except in the case of very fine slimes. Each double table requires about $\frac{1}{4}$ indicated H.P.; one man

can attend to two double tables without any difficulty. The capacity of a double table is about $3\frac{1}{2}$ to 5 tons per 24 hours; the consumption of water is considerable, being altogether about 0·5 to 0·8 cubic foot per minute, three-fourths of which amount constitutes the stream of clear water. In more modern forms the tables have been made of planished sheet-iron, of slate, and of plate-glass, whilst metal has been substituted for wood in nearly every part, the supporting frame being light castings, the springs steel, &c.

An end-shaking table has been largely used in Australia, where it is generally known as the Halley table. That shown in Fig. 93 is fairly representative of this class of machine. It was manufactured by Messrs. Appleby Bros., for use on the Gold Coast of Africa, where it is said to have done good service. It consists of a table suspended like the Rittinger table, but moved in the direction of its length by a cam (usually a three-throw cam, such as is shown separately on a larger scale above the sectional elevation) against a strong spring, which pushes it back with a sharp jerk against a bumping-piece, the spring being at the lower end of the table, and the jerk, therefore, upwards. When pulp is fed into this table, it is subjected to two opposite forces: that of the stream itself tending to drag the particles downwards, and that of sucessive jerks tending to throw them upwards. The heavier particles are more difficult to move by the current of water, whilst they acquire greater momentum from the impulse of the jerk. Accordingly the barren sands tend to travel down the table and are discharged at its foot, whilst the sulphurets travel up the table and are discharged at its head, or, as in the table shown, accumulate in a special division of the table,

whence they are removed by hand. These tables are simple and easy of construction, not expensive, and do

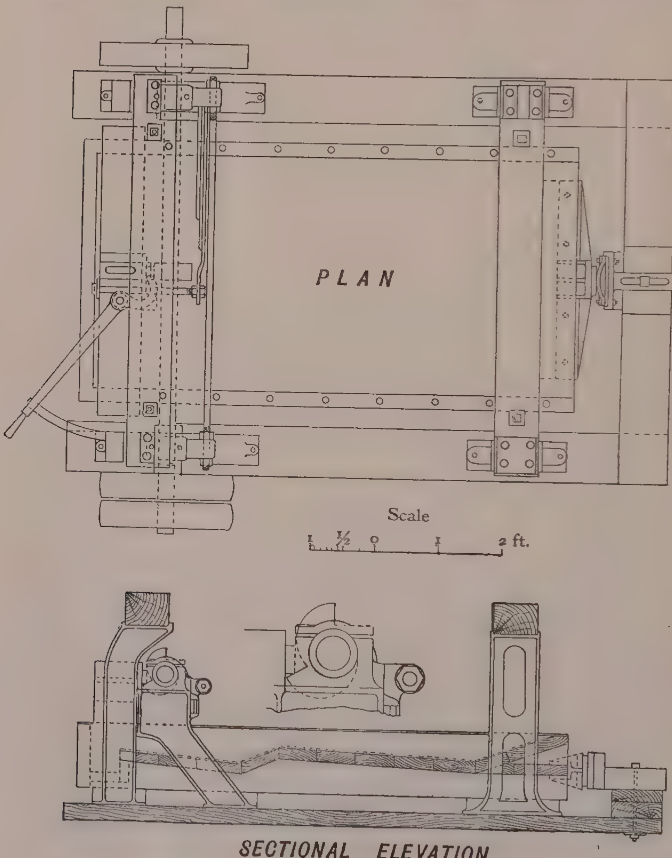


FIG. 93.

fair work except on fine slimes. The inclination of the table must be adjusted according to the character of the

pulp to be treated, and the nature and proportion of the concentrates obtained from it. The usual speed is about 200 blows per minute, and one machine about 4 feet by 8 feet will treat all the pulp produced by a five-stamp battery.

The Gilpin County bumper, or the Gilt Edge concentrator, as it is often called, is practically identical with the last-described machine, except that it is continuous-acting, discharging the concentrates at the upper and the tailings at the lower end, and that it is built of iron instead of being largely of wood. These machines are built in pairs and are run at 120 to 150 blows per minute; a double machine 7 feet long and 3 feet wide will treat 15 to 20 tons of tailings, averaging say 10 per cent. of concentrates, in 24 hours.

The Hendy Concentrator may be looked upon as a rather widely and divergent modification of the shaking-table. It consists of a shallow iron pan 4 feet in diameter, supported on a vertical shaft in the centre, and made to oscillate back and forth by means of cranks on a shaft at one side and joined by connecting-rods to the periphery of the pan. The pan has an annular groove at its outer edge about 2½ inches wide and deep, and receives about 220 sharp oscillations per minute. The machine is, in fact, a kind of circular shaking-table. When in operation all the particles tend to move downwards and to accumulate at the circumference, the heavier sulphurets occupying the lower portion and thus collecting in the annular groove. This groove is fitted with a gate by means of which the discharge from it can be regulated.

Each machine is guaranteed to treat five tons of tailings per day of twenty-four hours, but a larger amount, up to eight tons, can be put through, although in that case

some of the sulphurets may be lost, and the products will not be equally clean as when a smaller amount is being treated. A ten-stamp mill will therefore require five or six Hendy concentrators. The weight of each machine is about nine cwt. and its cost about $300 (£60 sterling), or say half the price of a Frue vanner. These machines are capable of doing fair work, except on fine slimes, and were at one time very extensively used on the Pacific Coast. They have, however, of late years, been practically replaced by some form or other of belt vanners. A somewhat similar machine is the Duncan concentrator, which

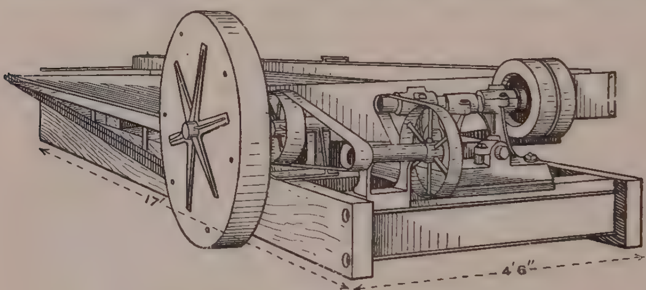


FIG. 94.

has a wrought-iron pan instead of a cast-iron one, its action being in most respects an imitation of that of the Hendy. It has, however, never come into public favour.

The Wilfley Table.—This concentrator is one of the most recent of this class of appliances, but has already met with marked success. Its general appearance is shown in Fig. 94; it consists of a flat wooden table, 16 feet long and 7 feet wide, covered with linoleum, upon which are nailed a series of strips of wood, gradually increasing in length from the back to the front of the

machine, and gradually tapering to nothing from a depth of about $\frac{3}{8}$ inch at the motion end. The table slopes upwards about $\frac{1}{2}$ inch from the motion end, and also slopes forward from the back; the amount of this latter inclination can be altered at will, according to the nature of the material that is undergoing treatment. The table is moved by an eccentric combined with a link and toggle, so as to have a quick forward and a slow backward motion; a spring keeps the table close against the motion the whole time, so that there is no shock or bump properly speaking. The pulp to be concentrated is fed on to the table from a head-board near the motion end for a width of some 3 feet; the rest of the table receives clear water only. When the table is in operation, quartz and other minerals of low specific gravity are carried by the stream of water down the table in a practically straight line; heavier bodies sink below the edge of the riffles, are thus unable to escape straight down the table, and are hence gradually moved along it by the series of impulses to which they are subjected; they are only capable of being carried by the water current when they have moved clear of the riffles, hence a particle of mineral on this table moves in a direction that varies from nearly parallel with the length of the table to nearly transversely across it, according to its size and specific gravity, or, if particles of practically uniform size are alone considered, they move according to their respective specific gravities. Clean tailings run at once to waste, middlings are returned by a small raff wheel to the head-board, and the heavier minerals are discharged at different points of the table in accordance with the principle already stated. The great advantage of this table is that it makes a very

clear and distinct separation upon various species of minerals, it does not, however, do equally good work upon unsized pulp, hence it should be preceded by Spitzlutten or some other form of automatic classifier. A table is intended to treat about 30 tons of ore per 24 hours, but has exceptionally been found capable of taking up to 50 ton; in some American mills one table is put in to every ten head of stamps. It requires about 1 H.P. to run it, and should be driven at 240 three-quarter-inch strokes per minute. The supply of clear water required varies greatly with the character of the ore to be concentrated; it may be said to range from 5 to 20 gallons per minute. The greatest defect in the machine is probably the raff wheel arrangement for returning the middlings. The weight of the table is about 22 cwt. and its price about £90. The success of this machine has caused a very large number of others constructed on about the same principles to be put on the market; they all combine the features of a transverse shaking-table like the old Rittinger table, with transverse grooves or riffles upon its upper surface; much ingenuity has been shown in making these riffles oblique, sinuous, &c., but the mode of action is in every case the same.

Belt Vanners.—This class include some dozen or more different machines, all more or less closely resembling each other, and all being modifications of the first one, the Frue vanner, which may be looked upon as the typical belt vanning machine. The best known machines of this type are the Frue, the Embrey, and the Triumph vanners, and the Lührig belt concentrator. In the first three the current of pulp is parallel to the longitudinal direction of motion of the belt, in the last it is across this direction; the first-named has a side shake, the second and third an

end shake. The general principle is always more or less the same.

Frue Vanner.—This machine is now so well known that any detailed description would be superfluous. It is shown in perspective in Fig. 95. It consists essentially of an endless rubber belt about 4 feet wide, having flanges at either side. This runs over a series of rollers set in a frame so as to form a flat table about 12 feet long, which slopes at a slight incline at from 3 to 6 inches in the total length. The belt moves upwards at an average rate

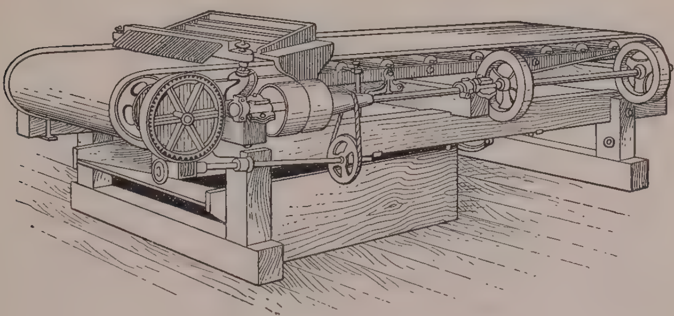


FIG. 95.

of 6 feet per minute. About 2 feet 6 inches from the upper end the pulp is discharged on to the belt with great uniformity by means of a distributor, or head-board, and about a foot above this, a number of fine jets of clean water play upon the belt. The frame carrying the belt is kept in oscillation at the rate of about 200 strokes per minute, the average length of the stroke being 1 inch. The machine is accordingly a form of shaking-table, in which the fixed surface is replaced by one moving slowly upwards. When the pulp is discharged upon the table,

each particle is subjected to two opposite forces—firstly, that of the stream of water, which tends to carry it downwards to the foot of the table; and, secondly, that of the motion of the belt, which tends to carry it upwards to the head, and the ultimate motion of each particle will be decided by the extent to which it is acted on by one or other of these forces. The entire pulp being kept in a state of agitation by the rapid oscillations of the table, the heavier particles tend to sink to the bottom of the layer of pulp. In the bottom layer of pulp, velocity of the water current is least, being retarded by friction against the belt, whilst the effect of the upward motion of the latter is a maximum, it being communicated directly to the particles in contact with it; in the upper layers, on the other hand, the effect of the upward motion of the belt is scarcely felt, whilst the velocity of the descending current is a maximum. Accordingly, the particles which occupy the upper layers of the pulp tend to move down the table, and those occupying the lower layers tend to move it up; in other words, the light barren particles of sand move downwards to the foot of the belt, whilst the particles of heavy sulphurets move upwards towards the head. At the same time large particles of barren sand, whose rate of falling through the pulp may be equal to that of smaller particles of heavier mineral, project further upwards from the table than do the latter, on account of their size. The former are therefore exposed to the action of a more rapid downward current than the latter, and are hence also carried slowly down the belt. This is the reason why this machine works well, even on unsized pulp, and why a very complete separation of the pulp into barren sands and valuable concentrates is effected by it. Of course it gives even better results if the pulp is first sized, or classified,

the different classes, being fed into different vanners. The work of separation is practically done between the points where the pulp falls on to the belt, and where the jets of clear water impinge on it, the action of the latter being to free the heavy concentrates from the last particles of sand by such a dilution of pulp as will tend to complete the separation. The application of this theory at once indicates the main points to which attention must be directed in regulating the machine.

In the first place, the machine must be set very substantially, so that only the proper motions may be communicated to the pulp, as other irregular impulses would produce cross currents interfering with the proper movement of the particles. The belt must be perfectly horizontal transversely, and the supply of pulp and of clear water must be perfectly uniform across it, so as to maintain in all parts of the belt an equal and sufficient depth of pulp to enable the separation by gravity to take place equally throughout it. It is found by experience that a depth of about $\frac{1}{2}$ inch of pulp on the belt gives about the best conditions in this respect. The velocity of upward travel of the belt must be such as to counterbalance the downward tendency of the finest particles of slimed sulphurets; accordingly, finely-crushed material demands a flatter belt and a faster speed of uphill travel than does more coarsely crushed pulp. Each one of these points must be carefully regulated by close observation for the particular class of ore under treatment, so as to enable the machine to work at its maximum capacity without loss of efficiency. Once adjusted, however, the machine requires but little further attention as long as the conditions remain unchanged. Special attention must be paid to this latter point. The vanners should be driven by a

special motor, independent from that which drives the mill and, above all, the rock-breakers, and furnished, if possible, with a sensitive governor. The counter-shaft should be well looked after, and all the driving belts kept in good order. The speed of the machines will thus be kept uniform. The clear water supply ought to come from a separate tank of ample size, and the water-main leading from it should be of large diameter. As long as the stamp-mill continues to crush the same class of material at a uniform rate, so long will all the conditions affecting the vanner remain unchanged, and the machine continue to work steadily and satisfactorily.

When the Frue vanner is in operation, the distribution of the pulp on the belt affords the best indication as to its working. All the conditions should be so adjusted that a small triangular patch of sand should show at each of the lower corners of the belt. These "sand corners" should not be too large, but must be well marked, and the two should be of equal size. Should they be unequal the fault will be found to be either in that the belt is not accurately level across, that the distributor is not doing its work thoroughly, or that some of the working parts have not been properly tightened up, so that there are other motions than the normal ones communicated to the belt; when once the cause of the want of uniformity has been determined, it is a very easy matter to remedy it. Too large a corner of sand shows that the pulp is too thick, whilst absence of any corner indicates that it carries too much water. The clear water supply should be only sufficient to keep the layer of pulp between the distributor and the water jets thoroughly and uniformly wet, and to prevent the formation of any channels through it, and once the proper rate of supply for this purpose

has been determined it should not be altered. If the pulp, as it leaves the mercury traps, is too dense, it must be diluted with clear water, either in the carrying launder or in the distributor; if it is too dilute, a concentrating **V**-box, such as is presently to be described, should be interposed. The capacity of the ordinary machine is about 6 tons per 24 hours, whilst a wider belt (6 feet in width) has been known to treat about 12 tons in the same time. The best practice seems to be to allow two vanners to every five heads of stamps. The amount of water required is about 0·2 to 0·4 cubic foot per minute in the pulp, and about half as much additional for the clear water jets. The entire machine weighs 21 to 22 cwt. and costs $575 (£125). It requires $\frac{1}{2}$ I.H.P. to drive it. One man can readily look after sixteen of these machines. After the belt has passed the head of the machine, it is bent down and passes into a tank, where the sulphurets are washed off it; the sulphurets accordingly accumulate in these washing tanks, and have to be taken out from time to time; this is best effected by an ordinary long-handled shovel. The overflow from these concentrate tanks always contain a good deal of slimed sulphurets in suspension, and these are usually the most valuable part of the concentrates. Hence the overflow from these tanks should be made to pass through a series of settling-boxes, where these finely divided sulphurets are deposited. The settling-boxes are cleaned out from time to time, usually when the mill clean-up takes place. For a small battery, having, say, not more than eight vanners, these are best placed in a single row. A larger number is best arranged in a double row, pointing away from each other; that is to say, with the head ends nearest the centre line of the building, with a space of about 5 feet between them.

Along the middle of the vanner house there should be a narrow-gauge track upon which run cars, which serve to carry the concentrates to their destination outside the vanner house, the attendant shovelling the sulphurets from the respective tanks into these cars.

The pulp, as it comes out from all the amalgam traps, should be run into one main launder, and thence supplied in separate launders to each vanner, every launder having a small gate by which the supply of pulp is regulated. These launders are best triangular in section, with a grade of not less than $\frac{3}{4}$ inch to the foot.

It is obvious, from the description above given of the principles of the vanner, that a belt with a rough or corrugated surface must act more effectively than one with a smooth surface, as it must increase the tendency of heavy particles to work upwards. Corrugated belts have been introduced, and the new machines thus produced seem to give in some respects better results in that their working capacity is greater than that of the old form, one machine being sufficient to treat the pulp from each five-stamp battery. These "Improved Frue Vanners" are manufactured by Messrs. Fraser, Chalmers, and Co., Limited, at a cost of \$835 (£170), their weight being also slightly greater than that of the old type (about 23 cwt.). When pulp is sized before going to the vanners, it seems preferable to treat the coarser portion upon corrugated, and the finer upon smooth belts.

As has already been said, the Frue vanner may be taken as a type of all vanners, and the above-given description applies practically to all of them, the differences in detail of construction not meriting separate description; these mostly refer to the character of the motion, the principle remaining always the same. End-

shake machines require to be run at a rather higher speed than does the Frue vanner, 230 blows being about the average. Amongst the newer vanners the Woodbury and Hendy may be mentioned, but so far none seems to be preferred to the Frue, which is a deservedly popular machine.

The Lührig vanner (Fig. 96), which is practically identical with the Stein-Bilharz machine, is essentially different from the above, inasmuch as the direction of the belt is at

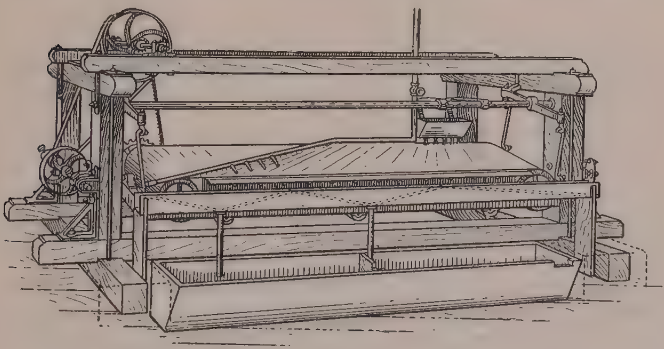


FIG. 96.

right angles to that of the other machines. The belt is horizontal in the direction of its length, but inclined some 4° across it, receiving impulses in the direction of its travel. It is usually about 3 feet 6 inches wide and 12 feet long, but the dimensions are often varied; the rate of travel is about 8 or10 feet per minute, and the number of impulses about 180, the length of stroke being from $\frac{1}{8}$ to $\frac{3}{4}$ inch. The pulp is fed on to the belt from a head-box placed parallel to the length of the belt close to the motion end, the rest of the belt receiving clear water only. The

machine thus resembles a Rittinger side-shake table or a Wilfley table, in which the table itself is continually advancing in one direction. It has been much improved in details of construction in recent years, and is now a good reliable machine. It will treat from 3 to 8 tons of tailings per 24 hours, takes $\frac{1}{3}$ to $\frac{1}{2}$ H.P. to drive it, and requires about 5 gallons of clear water per minute in addition to that in the pulp. Its chief advantage over the vanners already described is that it will make several grades of concentrates, like all side-shake tables, when working on mixed ores; this property is very valuable in general ore-dressing practice, but in gold milling, where clean tailings and clean concentrates alone are required, is of far less importance.

Sizing.—Of late years the advisability of sizing the pulp that comes from the stamp mill has been obtruding itself more and more upon the notice of mill men. This method has the advantage that comparatively coarse crushing can be used; it has already been pointed out (page 147) that only a small percentage of the total pulp is crushed to the maximum size allowed to pass through the mesh of the battery screens. It is an easy matter to separate this small proportion, which may be too coarse to admit of proper separation of the gold, and either return it to the stamps or treat it in separate grinding machinery, whilst the rest of the pulp can be classified into separate sizes, each of which can be treated on separate suitable concentrators. For this purpose one of the simplest classifiers is the V-box (German "Spitzkasten"). This has not undergone any substantial improvement since it was originally designed by Rittinger some thirty years ago. Its principle depends on the fact that if the velocity of a current of water, carrying particles of vary-

ing sizes and specific gravities, be decreased in definite successive ratios, a definite number of these particles will be deposited, corresponding to the decrease in velocity of the current. It has already been pointed out that a smaller particle of heavier material will fall through a column of water at an equal rate with a larger but lighter particle, when the sizes and specific gravities bear a definite ratio to one another, and the particles deposited at each diminution of the speed of the current will accordingly consist of such particles as will fall through equal spaces in equal times, the entire pulp thus undergoing successive classifying. The diminution of the velocity of the current of pulp is brought about by allowing it to flow through a series of boxes of successively larger sectional area; in each such box particles are deposited, whose size is a function of the area of the box, and nothing more than this is required to effect the sizing of the pulp. In order, however, that the discharge of the sized pulp may be effected automatically from each box, these boxes are made in the shape of an inverted pyramid, having a small aperture at the apex through which the sized pulp may escape. A series of four such pyramidal boxes is shown diagrammatically in Fig. 97, which has been taken from Rittinger's book. The first box should always be of such a size as to give a width of 0·1 foot for each cubic foot of pulp passing through it per minute. The length of the box is of less importance than its width, and may for the first box be taken at about three times its width. The widths of successive boxes should increase in geometrical progression—that is, in the proportions of 1 : 2 : 4 : 8, &c.—whilst the length may be in arithmetical progression. The angle of the pyramid should be at least 50° to the horizontal, in order that

no particles may settle on the sides; this pitch of 50° is given throughout to the transverse walls of the box, the longitudinal walls being best made vertical down to such a depth as will admit of a regular square pyramid terminating the box. For the larger size of box it is as well to make the point consist not of one but of two or more pyramids (Fig. 98), so that the angle of 50° may be maintained without unduly increasing the depth of the box to such an extent as to make it unwieldy. Successive

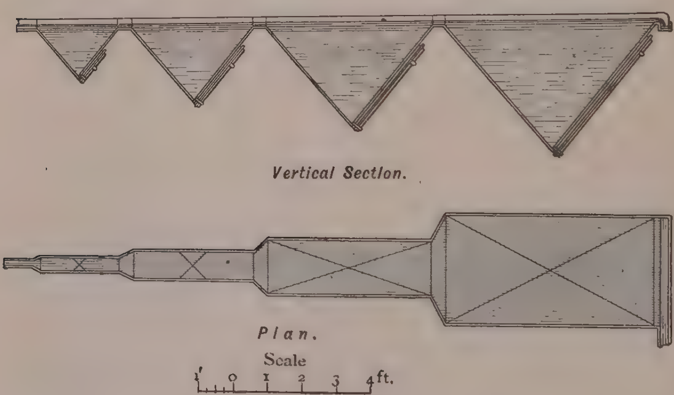


FIG. 97.

boxes may be connected by short carrying launders, which discharge the pulp on to a distributor, and thence into the box; it is as well to place a board on edge across the box, a short distance from the side at which the pulp enters, dipping a few inches below the surface of the pulp in the box, so as to break up any eddies or local currents across the box. The amount of water discharged with the sized pulp depends primarily upon the size of the aperture at the apex. In order, however, not to be

obliged to make this too small, a syphon discharge is employed, which, by bringing the point of actual discharge nearer the surface of the pulp in the box, diminishes the velocity of the issuing current, and hence delivers less water with the sized pulp. This is accomplished as shown by means of a rising pipe applied outside the box, and made of such a length that the point of discharge shall not be more than about 2 feet below the surface of the pulp for the fine slimes, and about 3 feet for the

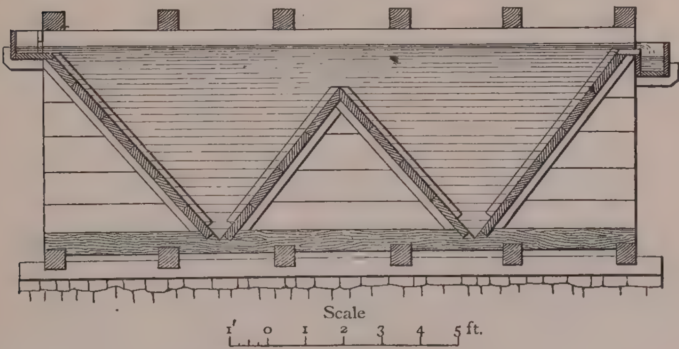


FIG. 98.

coarser material. The diameter of this syphon pipe will mostly vary between $\frac{1}{4}$ inch and $\frac{3}{4}$ inch, according to the amount of pulp to be discharged. This pipe must be fitted with two valves or plugs, one at the apex of the pyramid to be used in emptying the box, or when it is desired to flush out any accumulation of sand from the bottom of the box, and the other at the mouth of the syphon pipe to regulate the flow. In a few exceptional cases, where it is desired to obtain a product with very little water, this may be accomplished by keeping the valve

shut, and only opening it intermittently. The continuous action is, however, almost always preferred. Provision should be made to enable the pipe to be readily cleaned out should it become stopped up. This method of sizing presents very many advantages; the plant is very cheap, and readily constructed by any ordinary carpenter, requires little or no repairs, and when once well started needs hardly any attention. When the battery, and consequently the flow of pulp, is stopped, the latter should be replaced by a flow of clear water to prevent any settlement of sand and consequent obstruction in the discharging syphon. Such boxes are now often replaced by sheet-iron cones.

Another and perhaps preferable form of this same machine is the "Spitzlutte." This, as now constructed, consists of a series of pyramidal or conical boxes, which may be made of either wood or sheet-iron, carried in a suitable frame. Within these boxes are suspended smaller similar ones, the pulp being led into the interspaces between the two. The delivery valve is arranged like that of the "Spitzkasten." At the apex of the box there is introduced an upward jet of clear water. Upon the strength of this flow of water and the area of the annular space between the two casings depends the rate of deposition of the sand carried by the pulp, and sizing is effected as in the previous apparatus. A neat form of "Spitzlutte," constructed entirely of iron, is shown in Fig. 99, this being the design of the Grusonwerk Company of Magdeburg.

There are several other means of sizing tailings, all depending upon the same principle, or variations of it, but the V-box and "Spitzlutte" are probably the most convenient of all of them. When the tailings are thus

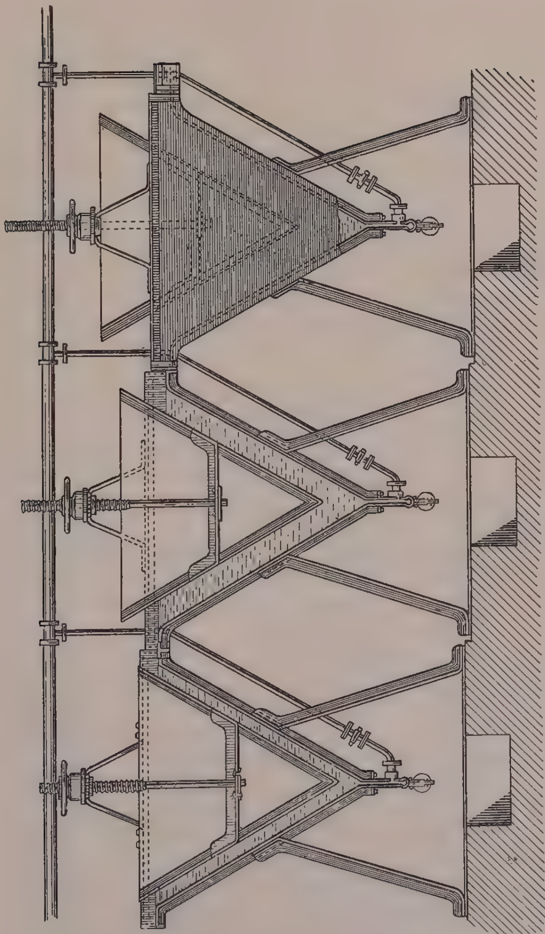


Fig. 99.

sized, such particles as are too coarse and need further crushing may, as already said, be re-treated either in the battery or in some form of grinding machine; the coarser portions of the pulp which do not require re-grinding may be treated either on belt vanners, or on some cheaper form of concentrator, generally a shaking-table, or even on fine jigs when very coarse, whilst the belt vanners are employed solely on the finer grades of material. It has already been stated that the belt vanner is capable of doing thoroughly good work on unsized material, but at the same time it is always an advantage, and better results will be obtained, when it is only required to treat sized sand, as it is easier to adjust it with greater accuracy when the size of the stuff it has to treat is confined within definite limits than when this varies through a very wide range. Moreover, this system admits of the employment of a cheaper form of concentrator for a large bulk of the tailings to be treated, and hence more perfect and more economical concentration can be obtained when the system of sizing is practised. Until the time shall come when the entire bulk of the tailings will be treated by a continuous chemical process (and it may not be very long before this is accomplished), the system of sizing before concentration represents the most important improvement now available in the actual practice of producing concentrates; it is, moreover, finding a further application in that it will also automatically and cheaply separate out slimes of any required degree of fineness, and is thus used for preparing tailings for chemical treatment; fine slimes oppose the greatest difficulty to such treatment, because they pack in amongst the coarser sands, making an almost impervious mass

which practically defies percolation. The separation of slimes is now accordingly a preliminary step, that is almost always resorted to when battery tailings are to be submitted to cyanidation.

Elmore Process.—The Elmore process of oil concentration depends upon the selective surface attraction that appears to subsist between certain oils, especially, it would seem, mineral oils; and the great majority of metals, metallic sulphides, arsenides, and similar compounds, whilst oxides and earth minerals suspended in water are quite indifferent. If, therefore, battery pulp containing, for example, quartz, native gold, and pyrites, be stirred up with oil, the latter will take up the gold and pyrites, but not the quartz. If the oil is present in sufficient quantity, its specific gravity, even when charged with these minerals, will be considerably below that of water, so that it will form a layer floating upon the water, the quartz, of course, falling to the bottom of the containing vessel.

In practice the process is carried on by passing the pulp successively through three horizontal mining cylinders about 10 feet long and 3 feet in diameter, making about six revolutions per minute; there are special ribs inside the cylinders, which causes the pulp to trend forward steadily, at the same time mixing it with the oil. The mixed pulp and oil run into a V-box, not unlike a mercury trap, which separates the oil from the pulp, a second similar separator removing the last traces of oil from the latter. The oil charged with mineral goes to a solid basket or drum centrifugal machine in which the bulk of the oil is removed, to be used over again. The concentrates are transferred to an open basket centrifugal machine in which most of the remaining oil is recovered,

the concentrates being now ready for further treatment. Such a unit of plant can treat about 25 tons per day and requires about 8 H.P., together with a supply of about 25 tons of oil.

The process is simple and cheap, the main expense being the cost of the oil adhering to the final concentrates; these carry about 3 per cent. of their weight of oil, so that the loss of the latter depends upon the percentage of concentrates present in the ore. The oil used is the coarse residual oil left in the rectification of petroleum, the cost of which is very low. Any oil will serve as long as its specific gravity is low enough and it is neither too limped nor too viscid.

The special advantage of the process consists in the fact that it is practically independent of the specific gravity or of the size of the particles to be separated, and is hence especially applicable to the finest slimes; it is well known that many gold-bearing minerals, some of which often carry the greater part of the gold values, *e.g.*, copper pyrites, tellurides of gold, &c., are excessively brittle, and therefore forms such extremely fine slimes when worked that it is practically impossible to save them by any method of mechanical concentration. It is probable that the Elmore process can be applied with advantage in such cases. As the loss of oil is proportional to the percentage of concentrates present, such concentrates as can be collected by mechanical means should be removed first, and the Elmore process employed to recover the remainder. The best arrangement would seem to be to send the pulp after leaving the wells on to Wilfley, or some allied form of tables, on which the coarser concentrates will be taken out, the pulp going from these to

the Elmore plant. The process is quite novel, and has not yet been submitted to sufficiently exhaustive practical tests, but it seems to promise well, and several gold mines are now erecting Elmore plants. The cost of a plant to treat 100 tons per day is about £2,000, and the cost of treatment should be between 1*s*. and 1*s*. 6*d*. per ton.

CHAPTER XII

TREATMENT OF CONCENTRATES—AMALGAMATION—CHLORINATION—SMELTING—CYANIDATION

Treatment of Concentrates.—After the auriferous sulphurets have been obtained in the form of concentrates, they have to be further treated to extract from them the gold which they contain. At one time this used to be done almost entirely at the mill by some method of amalgamation, which extracted a varying proportion, but never the whole of the gold. Of late years, however, the treatment of sulphurets has been placed on a more scientific basis, and is now rarely a part of the mill man's duty, concentrates being either sold according to assay to outside works which treat them, or else, in the case of sufficiently large mills, being handed over to a separate department for the extraction of their precious contents. In either of these latter cases their treatment no longer falls within the province of the mill man, and it is sufficient for the latter therefore to understand merely the principles which underlie their after treatment, in order that he may furnish a product in the best possible condition for the processes it has to undergo.

There are three main methods of treating concentrates:

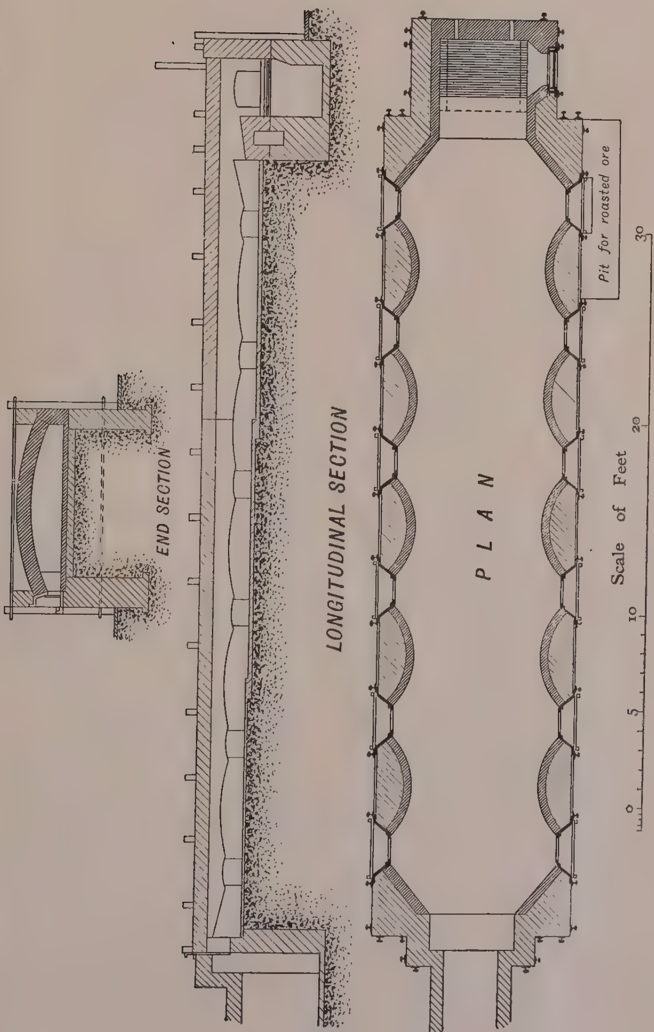
A. In the mill—by amalgamation.

B. Outside the mill—

1. By wet methods.
2. By smelting.

Amalgamation.—It has already been pointed out that it is by no means certain even now in what form gold occurs in concentrates. It is certain that some of it is in the form of free gold in minute lamellæ sealed up so to speak in the particles of pyrites, and that some of it is chemically combined in tellurides; it may also exist in other forms, *e.g.* as coated gold, as allotropic gold difficult of amalgamation (except under special conditions which tend to convert it into amalgamable gold), and as chemically combined gold, or gold so intimately associated mechanically with other minerals as to present great resistance to a comparatively feeble chemical agent, such as mercury. Whatever theories be held on these points, the facts are, at any rate, that when concentrates are ground up in intimate contact with mercury, varying proportions of gold are extracted, the proportion depending *inter alia* upon the nature of the ore as well as on the fineness of the grinding and the prolongation of the operation. If, however, the concentrates are first calcined before being amalgamated, practically the whole of the gold can in most cases be extracted by amalgamation, and it is reasonable to suppose that the whole could be extracted were it possible to completely calcine the sulphurets without causing fritting by incipient fusion. As a general rule, concentrates are only treated by methods of amalgamation when the mill producing them is situated in so remote and inaccessible a district that the cost of conveying them to the nearest market is greater than the excess of the profit derived from their sale over their yield when amalgamated, where the cost of carriage augments greatly the price of the chemicals

required for the application of chemical methods, and where the amount produced is so small as not to warrant the erection of separate works for their treatment; this system is hence confined to small and comparatively imperfect installations. It is very rare indeed that sulphurets are calcined before amalgamation, although the advantage of so doing is in most cases very decided. In many instances the cost of a calcining furnace is looked upon as prohibitive, although there is no doubt that it would frequently pay for itself in a very short time. Even partial calcination is of advantage, such as may be carried out by forming heaps upon a thoroughly smooth and even floor, say of clay rammed hard, or of bricks or tiles set in clay puddle. Upon this floor a layer of brushwood about 1 foot deep is piled, and upon this a layer of sulphurets of about the same depth. The pile is surrounded on three sides by low walls which may be permanent or movable; a draught is established through the centre of the pile from end to end, and the brushwood fired. By suitably protecting the pile from the wind and by damping it down when the fire becomes too fierce, a steady combustion of the brushwood may be secured, which is propagated to a great extent through the layer of concentrates by the combustion of the sulphur contained in them, and by this means a partial calcination is obtained, which, although far from perfect, greatly facilitates the liberation of the gold. When the quantity of concentrates is sufficient, and amalgamation has nevertheless to be adopted, it is always worth while to build a small calcining furnace. A furnace suitable for the calcination of sulphurets is shown in Fig. 100. On the other hand, pan amalgamation is very rarely applied to clean sulphurets, blanketings being frequently treated



END SECTION
LONGITUDINAL SECTION
PLAN
Pit for roasted ore
Scale of Feet
0
5
10
20
30

FIG. 100.

in this way, and these will probably not contain more than 10 per cent. of sulphurets and at times not half that amount, the remainder consisting of the coarser portions of the quartzose sand. Under such circumstances it would naturally not pay to adopt calcination unless the blanketings were first further concentrated. In some mills it is usual to allow the concentrates to weather for a longer or shorter time before treating them in the pan. For this purpose they should be spread over a level floor in the open air, and watered and turned over from time to time. The slow natural oxidation which is thus set up is decidedly beneficial in rendering the gold more readily amalgamable. Sometimes, however, the tailings, as they leave the stamp-mill, are run direct into an amalgamating machine, where they are rubbed up with mercury. Whenever this method or that of amalgamation after slow weathering is employed, it is of less importance that quicksilver and amalgam should be completely removed from the concentrates, than when other methods of treatment are employed, as any amalgam that may be present is recovered in the process of further amalgamation. In fact, the use of mercury traps may in this case be dispensed with, or else their contents may simply be mixed with the concentrates, or transferred direct to the amalgamating pans.

The method of amalgamation is, however, even at the best, a crude and wasteful method, which will probably not long be practised anywhere, except in the rare cases already indicated. It is therefore needless to enter into any great details of a process which no scientific mill man is anxious to see survive the introduction of the far better methods which are now available to replace it.

Pans.—The number of pans that have been invented for

amalgamation is very great indeed. They may all, however, be reduced to two types: the ordinary Californian pan, which originated in America, but is by no means confined to that continent, and the Berdan pan, used almost exclusively in the Australian colonies.

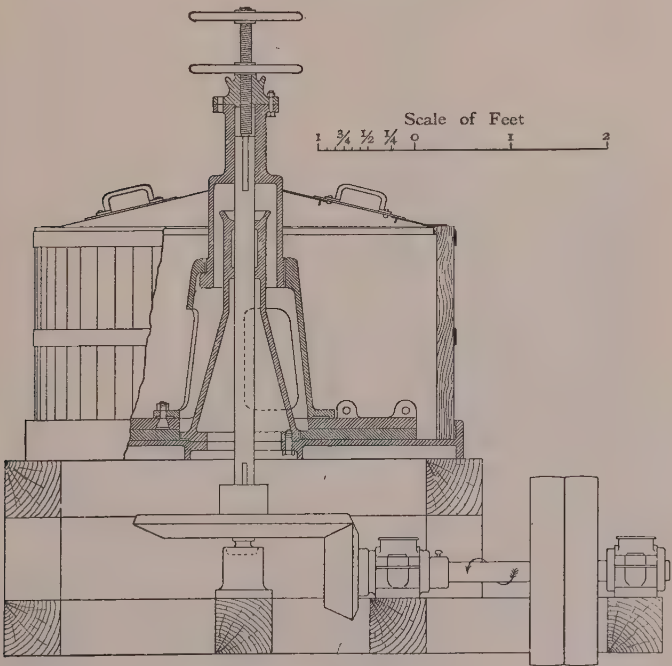


FIG 101.

Californian Pan.—The simplest type of this is shown in section in Figs. 101 and 102, which differ merely in some trifling details such as the methods of securing the shoes and dies. The former is constructed by Messrs. Bowes

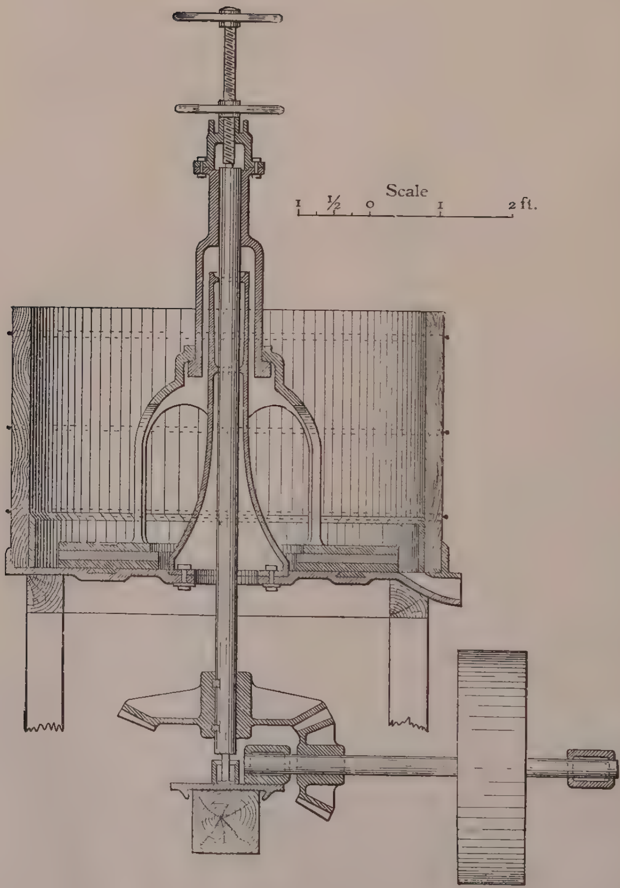


FIG. 102.

Scott and Western, Limited, and the latter by Messrs. Fraser and Chalmers, Limited. Other modifications relate almost entirely to alterations in the shape of the muller and pan bottom, and to the arrangement of wings on the sides, the former having for object a more efficient grinding action, and the latter a more thorough circulation of the pulp. The plain pan as shown, generally known as the "Combination" pan, is, however a very fairly efficient machine. These pans are mostly between 4 and 6 feet in diameter, 5 feet being a very usual size. This pan has a cast-iron bottom, whilst the sides are made of wood held together by iron hoops, thus combining strength and lightness. In order to protect the bottom from excessive wear and tear, dies are fitted to it by means of dove-tailed joints, these dies being made usually of hard cast-iron; sometimes white iron is employed, but heavily chilled iron is better. There are usually eight dies to a pan. The muller consists of an annular casting, driven by a central spindle, which passes freely up through the conical centre of the pan; this spindle rests on a step which carries a case-hardened steel button, the bottom of the spindle being rounded off, and also case-hardened to diminish friction; sometimes this bottom end of the spindle is made removable, so that it can be renewed when worn. A feather fits in a groove in the upper part of the spindle which drives the muller, at the same time allowing the latter free movement in a vertical direction. The muller can accordingly be raised or lowered by the screw and hand-wheel shown at the upper end of the spindle, a lower hand-wheel carrying a jam nut, which keeps it at any desired height. The lower grinding face of the muller is protected by dies similar to the shoes of the pan. This pan is fairly portable, its total weight, set up, being but

little over 3 tons. In starting the pan a charge of about 1 to $1\frac{1}{2}$ tons of sulphurets (for a 5-foot pan) is shovelled or dumped into it, the muller being raised out of the way; it is, however, advisable to start the muller rotating before the charge is thrown into the pan, so that it may all be gradually set in motion. Enough water is run in to make a stiff pulp, the degree of dilution being the most important factor in the successful operation of the machine. The pulp must be stiff enough to retain globules of quicksilver in suspension throughout it, but not so stiff as to impede too greatly the movement of the muller, or to prevent free circulation of the pulp. Generally speaking, the pulp should two-thirds fill the pan; it must be remembered that when the muller is revolving, the motion of rotation communicated to the pulp causes the latter to rise considerably up the sides of the pan. It is advisable to have a light wooden cover made in two halves to fit on the pan when it is in operation, and which should be put on as soon as the pan is charged. The muller having been started at its usual speed of 60 to 70 revolutions per minute, it is brought down by the hand-wheel until the shoes are just in contact with the dies, and it is kept running in that position grinding the sulphurets as fine as may be thought advisable. Usually this stage of the process is continued for two or three hours. The requisite amount of quicksilver is then poured in; the amount will vary with the richness in gold of the concentrates, and the quantity of the latter worked at each charge. Generally speaking, about 50 lbs. may be used. It should be throughly purified and alloyed with sodium exactly like the mercury used in the stamp-mill. The muller is kept revolving, but may be slightly raised at this stage of the process. A little cyanide of potassium

is sometimes added with the quicksilver, but there seems to be no definite object served by this addition, which is probably injurious rather than beneficial. A little alkali may be added if there is any fear of the charge in the pan having been contaminated with oil or grease. The amalgamation is continued for some time longer, usually for about six hours altogether.

When the quantity of blanketings or concentrates to be treated is sufficient to keep two pans in operation—that is to say, when it amounts to more than some 6 tons for 24 hours—it is usual to employ a settler. This is merely a larger and lighter pan, usually 7 to 8 feet in diameter, having four to eight arms instead of a continuous annular muller; these arms are usually furnished with wooden shoes, and the bottom is rarely fitted with dies, as the wear is not great. The settler is provided with a series of discharge pipes with plugs, up one side, so that its contents may be drawn off at any desired height. A special syphon tap, by means of which mercury and amalgam may be drawn off from the bottom of the pan, is also provided; there are usually one or more grooves cast in the bottom of the pan in which the mercury and amalgam collect, and which communicate with the syphon tap. The contents of the pan are run out into the settler and considerably diluted with water, so as to enable the quicksilver to fall freely through the pulp. The spindle carrying the arms is set revolving at about 15 to 20 revolutions per minute, until the mercury and amalgam have fairly settled, when the top hole is unplugged and the contents drawn off to that depth, the spindle being kept revolving slowly; plug after plug is thus opened until all the slimes are run off, when the settler is ready to receive a fresh charge. Usually a charge can

be worked off in 3 to 5 hours. The slimes should be run through a sluice lined with amalgamated copper-plates to catch any particles of amalgam which they may still carry. When the amount of concentrates to be treated is not so large as to require the above plant of two pans and a settler, one pan may be made to do all the work by furnishing it with a series of plug-holes, and by providing the counter-shaft that drives it with a cone-pulley so as to enable the speed of driving to be varied. A small charge of concentrates, say 1 ton at the outside, should be treated at a time, the pan being rather less than half full. When amalgamation has proceeded for a sufficient length of time, the muller is raised up half an inch or so from the bottom, the pulp is largely diluted, and the muller is run at about 12 revolutions per minute till all the mercury is settled, when the pulp is drawn off just as when the separate settler is used. The progress of both amalgamation and settling should be watched by taking out samples from time to time, and "panning up" or "horning" them so as to see in what condition the pulp and quicksilver are. The power required to drive a 5-foot pan is usually between 3 and 4 I.H.P., and a settler 2 to 3 I.H.P. Care must be taken in examining covered pans not to bring a light near them as soon as they are uncovered. An explosive gas has been known to accumulate inside the upper part of covered pans, and several accidents have been caused by its ignition. It is not difficult to understand how hydrogen can be generated during the operation of pan amalgamation, and the proper ventilation of amalgamating pans is a by no means superfluous precaution, which is, however, much too frequently neglected.

Berdan Pan.—Usual forms of this pan are shown in

Figs. 103 and 104. It consists essentially of a cast-iron shallow basin having a central cone, which is keyed firmly to a spindle set at an angle of about 15° to the vertical; usually a row of these pans is driven by means of bevel gearing from one counter-shaft. In the annular space between the central cone and the rim of the basin are either a couple of heavy cast-iron balls running loose (Fig. 104), or a couple of heavy drags attached by means of short chains to a portion of the frame through which the spindle passes (Fig. 103). The former drawing is from a pan manufactured by Messrs. Appleby Bros., and the latter from one by Messrs. Bowes Scott and Western, Limited. The balls or drags, by reason of their weight, maintain their approximate position, moving to and fro but slightly in the lower portion of the inclined annular space whilst the pan revolves, the usual speed being about 25 revolutions per minute. The pan is preferably lined with a liner piece of hard white or chilled iron, which can be renewed when worn out, as shown in Fig. 103. Pulp is fed in at the highest portion of the periphery of the pan, and as the latter continues to revolve it is ground between it and the weights (balls or drags) until fine enough to be carried by the stream of water over the lower edge of the basin, whence it flows off through suitable sluices. Mercury is fed into the pan from time to time. The process is thus a continuous one, and in so far it is preferable to the intermittent action of the Californian pan. The capacity of the Berdan pan depends very largely upon the degree of fineness of the pulp fed into it and the degree to which this needs to be ground, the latter being regulated by the speed of the pan and the amount of the water supply; an increase in either of these factors increasing the working

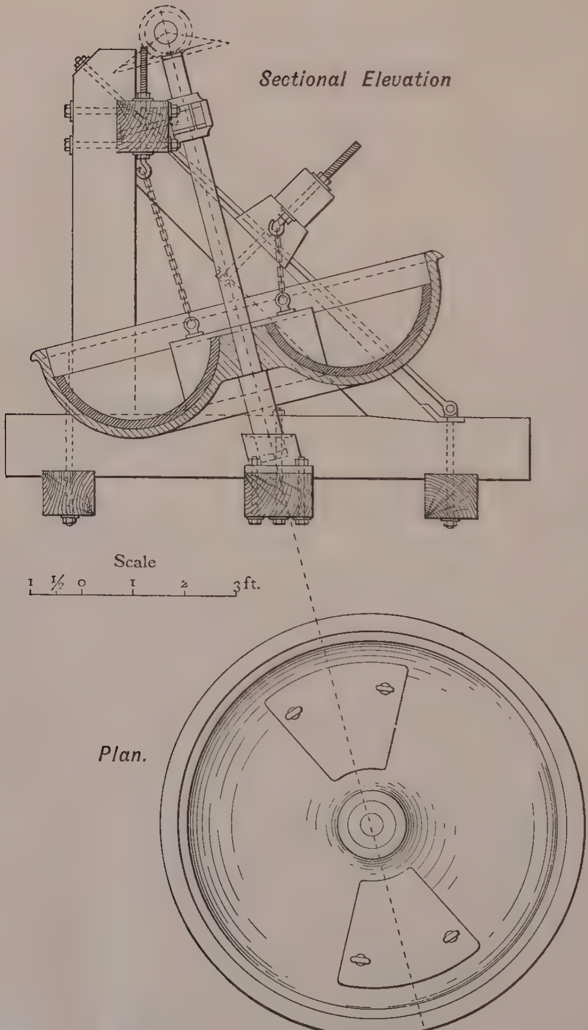


FIG. 103.

capacity of the pan by shortening the time during which the pulp remains in the pan. On the average its capacity may be taken as 2 tons per 24 hours; usually one pan is supposed to take all the blanketings from 5 heads of

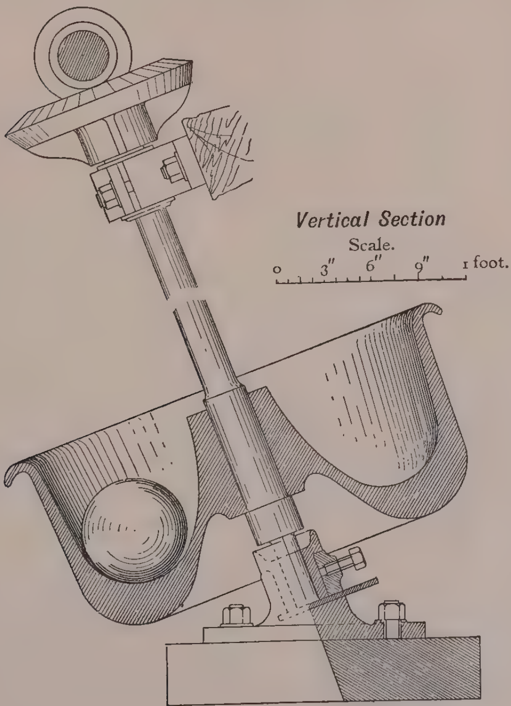


Fig. 104.

stamps. The upper portion of the sluices should always be lined with amalgamated copper-plates, and these may advantageously be followed by riffles.

When pan amalgamation has to be employed at all

the Berdan pan presents several advantages. It is a cheap machine, costing only about £50; it is continuous in its action, and needs practically no attention. The wear and tear is but slight, and only on cheap parts readily renewed. It does not absorb much power, about $1\frac{1}{2}$ I.H.P. being sufficient to drive each pan. On the other hand, it requires rather more water than does the Californian amalgamating pan.

Amalgamators.—A vast number of machines have been designed under this name, all of which have for their object the treatment of tailings by rubbing them up, or otherwise bringing them into intimate contact with mercury. It need only be said that if there is no scientific justification for the amalgamation of concentrates, there is still less reason why such a process should be applied to tailings. No amalgamating appliance can save gold that cannot be equally well caught in the Californian stamp-mill, if the latter is properly handled, and such machines are only used where milling is imperfectly performed. In Australia, Berdan pans are sometimes used in this way, and in Hungary the Laszlo amalgamator, a kind of small continuous-acting pan, is in considerable favour.

It is only necessary to conclude the subject of amalgamation of concentrates and tailings by repeating that for gold ores it is practically a thing of the past. In the Witwatersrand district, where the treatment of concentrates and tailings is being rapidly elevated to the dignity of a special science, out of 40,000 ounces of gold extracted from concentrates, under 300 ounces were obtained by pan amalgamation in 1892, since which date this method seems to have been entirely abandoned.

Chlorination.—This process was originally invented by Percy and Plattner independently about the year 1848,

and is still carried on according to the principles laid down by the latter, although all its details have undergone much modification, and have become the subject of numerous patents; there are thus some dozen chlorination processes in existence, and several in operation. The process depends on the fact already stated, that chlorine has a strong affinity for native gold, and readily combines with it, forming the soluble auric chloride (page 30). The solution containing the gold can be filtered off from the residue, which is now practically free from gold, and the gold in the solution precipitated. In the Californian modification of Plattner's process, which is on the whole perhaps still the best mode of carrying it out, the substance to be treated, properly moistened, is shovelled into a vat with a double bottom, the upper, false bottom being perforated and supporting a suitable filter. Chlorine gas is conducted into the space below the false bottom, and gradually rises till the vat is full of it; the lid is then luted on, and the whole left until the action is complete, when the soluble chloride of gold is washed out through the filter into other vats, where the gold is precipitated. Various precipitants, such as ferrous sulphate, charcoal, sulphuretted hydrogen, sulphurous acid gas, and copper sulphide have been used for this purpose. It is obvious from this brief description that the presence of any substance attackable by chlorine will cause a waste of that gas, which forms, under suitable conditions, the most expensive item in the treatment by chlorination. Among many other bodies, iron pyrites and most sulphurets are attacked by chlorine, whilst most peroxides, and ferric oxide amongst them, are not. It is therefore necessary to calcine the sulphurets produced in concentration as completely as possible in order to avoid loss from

this source, while calcination may possibly also serve to render the gold more readily attackable. Numerous types of furnace have been designed for the purpose of calcination, and the question of calcining without or with the addition of salt, and the losses of gold thereby incurred, have been very fully studied; the simple long-bedded calciner (Fig. 100) is still perhaps the favourite type of furnace. The chlorinating apparatus proper is the portion on which most of the patentees have exercised their ingenuity. Mears, Thies, and Rothwell in America, and Newbery and Vautin in Australia, have replaced the stationary vats by revolving barrels, into which the chlorine is either forced, or within which the chlorine is generated by suitable chemical means. Munktell has designed a method, used at Fuhlun, in Sweden, by which a dilute solution of chemicals capable of generating chlorine should be run over the ore contained in large vats. Various forms of filters have also been used, and a long list might be given of possible precipitants which have been proposed, but the broad principles of the chlorination process, with which alone the mill man need concern himself, are throughout the same. It is, in the first place, obvious that the sulphurets must, for the sake of economy, be as clean as possible. It would cost nearly as much to treat a ton of rough blanketings containing, say, 90 per cent. of barren quartz sand, and 10 per cent. of auriferous sulphurets as it would to treat a ton of the clean auriferous sulphurets, so that the cost of extracting a given amount of gold would be practically ten times as much in the first case as in the second. Concentrates intended for chlorination must, therefore, be as clean as the best type of concentrating machinery can make them; even if it were only for the saving

of freight from the mill to the chlorination works, this item would prove to be an important one. It must be remembered that highly fusible sulphides, such as those of copper and lead, have an injurious action in this process. On account of their low fusibility they tend to soften and frit and to cement together, so to speak, the partly-calcined particles of less fusible sulphides, so that properly complete calcination becomes impossible; thus not only is chlorine wasted, as already pointed out, but gold may be locked up in the semi-fused mass and, not being attacked by the chlorine, become entirely lost. Zinc sulphide is, on the other hand, favourable to a thorough calcination, as it will stand a high temperature in the calcining furnace without softening. If, therefore, a gold ore contains a high percentage of galena, it is not well suited for chlorination, but should preferably be treated by a smelting process, unless the galena is separated out from the other sulphides by special dressing methods, which its high specific gravity renders by no means difficult. The Lührig belt vanner may do good service in this direction. If the gangue of the gold ore be calcareous, even more than ordinary attention must be directed to producing clean sulphurets, and it will be better to accept the alternative of a small loss of the valuable materials rather than to risk their not being quite clean, seeing that lime has a great affinity for chlorine, and would thus tend to waste a great deal of this comparatively expensive reagent.

The table on page 400 shows the cost of chlorinating sulphurets in some of the principal establishments in North America.

The cost in South Carolina, as given by Mr. Thies, is determined upon roasted ore, 1 ton of which is equivalent to $1\frac{1}{3}$ tons of raw pyrites; the barrel process, as used

Localities and Methods.	Tons treated per 24 hours.	Labour.		Power and Water.		Fuel (Wood).			Chemicals.		General Expenses.		Total.				
		$	cts.	$	cts.		$	cts.	$	cts.	$	cts.	$	cts	£	s.	d.
1. Bunker's Hill Gold Mine, California, Barrel process . . .	2	4	75	0	50	$\frac{5}{8}$ cord @ $6	3	75	2	61	[b]3	00	14	61	3	0	11
2. Haile Mine, South Carolina, Barrel process	4	2	53	0	12	$\frac{1}{2}$ cord @ $1.50	0	75	0	48½	[c]0	15	4	03½	0	16	10
3. Plymouth Mine, California, Plattner's process	3⅓	[a]5	15	...		$\frac{21}{40}$ cord @ $2·45	2	23	2	02	[b]4	00	13	40	2	15	10
4. Amador Gold Mines, California, Plattner's process . .	3	4	00	...		...	3	00	3	00	3	00	[d]13	00	2	14	2
5. Sutter Creek Reduction Works, California, Plattner's process	4½	[a]6	66	...		$\frac{1}{2}$ cord @ $6	3	00	3	12	[b]3	70	17	48	3	12	10
6. Alaska Treadwell Gold Mining Co., Alaska, Plattner's process, 1892	12¾	5	34	...		$\frac{2}{7}$ cord @ $4.25	1	21	1	68	0	76	8	99	1	17	5½
7. Ditto, 1898	21	3	97	...		$\frac{2}{7}$ cord @ $4.25	1	21	2	64	...		7	82	1	12	7
8. Rapid City Mill, Dakota, Barrel process	40	1	20	0	29	$\frac{1}{6}$ cord @ $3.50	0	58	1	40	0	57	[e]4	04	0	16	10

[a] Includes superintendent's salary. [b] Includes assaying, repairs, supplies, insurance, taxes, water, interest on capital and other charges. [c] Includes superintendence and wear and tear only. [d] Does not include superintendence and office expenses. [e] Exclusive of superintendence.

there and in Dakota, is evidently much cheaper than the older type of vat process, but it must not be forgotten that the former requires a much greater capital outlay than the latter. At the Utica Mine, California, the cost of chlorinating by the Plattner process is said to be \$6.50 (27*s.*) per ton. The cost of barrel chlorinating in large works at Cripple Creek, including crushing, is said to be about as follows:—

	$
Labour, including salaries	1.34
Fuel for calcination and power . . .	0.70
Chemicals and supplies	0.72
Miscellaneous	0.77
Total cost per ton of ore . . .	3.53 (14*s.* 9*d.*)

The total cost of erecting the Sutter Creek works is stated to have been \$15,000, say about £3,000. Recently, however, such works have been erected more cheaply; in 1889, according to the eleventh U.S. census, the first cost in California of a chlorination works capable of treating 6 tons in 24 hours was between \$6,000 and \$7,000, and this may be taken as an accurate estimate for the present day. The same authority gives the total cost of chlorinating a ton of concentrates at about \$10, with an average recovery of 90 to 92 per cent. of the assay value (of gold) of the ore. The prices for custom chlorinating vary somewhat. The usual practice is to buy the concentrates outright at a certain percentage of their assay value—this percentage increasing with their richness—and deducting a fixed charge for chlorinating. The following is the scale of prices at the Sutter Creek Reduction Works and others in Amador county under the same ownership:—

For sulphurets assaying $50 of gold per ton and over,	90	per cent. of the assay value, less $20 per ton charges for working.
100	92	
200	83	
300	94	
400	95	
500	96	

Most customs chlorinating works in California, of which a considerable number exist, charge about $20 per ton for chlorinating, and guarantee a return of about 90 per cent. of the total gold contents of the ore, the percentage thus guaranteed increasing slightly with its richness; in fact, the scale of prices at the Amador works may be taken as applicable to the whole of California. In Utah the sampling works do not use a sliding scale like the above; the usual practice there is to pay for the gold at the rate of $12 per ounce, and to make a charge of $12 or more per ton, according to the character of the ore, for its treatment. When the concentrates contain a notable proportion of silver, this is paid for by special agreement. In Colorado, where chlorination and cyanidation works are treating the same ores, the prices paid are in some places about $19 per ounce of gold for less than 2 ounce ore, and $20 per ounce for ore assaying above 2 ounces, less $7 to $10 per ton for treatment. In Cripple Creek all the gold is paid for at the rate of $20 per ounce, less a charge per ton for treatment, the charge being as follows:—

	$	
For ore assaying under 15 dwt.	5.50	per ton
,, ,, between 15 dwt. and 1 oz.	6.50	,,
,, ,, ,, 13 ,, 1 oz. ¼ dwt.	7.25	,,
,, ,, ,, 1¼ ,, 1½ oz.	8.00	,,
,, ,, ,, 1½ ,, 2 oz.	8.50	,,
,, ,, ,, 2 ,, 3 oz.	9.00	,,
,, ,, over 3 oz.	9.50	,,

Reduction works near Nevada city buy concentrates at prices varying from 15 to 92 per cent. of their assay

value in gold, less \$18 per ton charges of treatment; when silver is present to a greater value than \$10 per ton, 60 per cent. of its assay value is paid for it. In Australia the usual charge for chlorinating is £3 per ton of ore delivered at the works, with a guaranteed extraction of about 95 per cent. The cost of chlorination in the Transvaal is also about £3 per ton.

Attempts have repeatedly been made to replace chlorine either wholly or in part by bromine; technically, the process leaves nothing to be desired, but the high process of bromine is a serious obstacle to economic success.

Smelting.—This is the most universally applicable process, and all concentrates, whatever their composition, can be treated by it; a large percentage of zinc blende is, however, objectionable, owing to the infusibility of zinc compounds. It must be remembered that smelting can only be practised when suitable ores for mixing are available, so as to produce a proper furnace charge, and hence it is never carried on at the mine, but only by customs smelting works, which buy the concentrates from various mills and mix them to suit their own convenience. The general principles are as follows: When argentiferous lead ores, such as galena, are smelted in a blast furnace, it is necessary to add a flux, of which oxide of iron is an essential ingredient, the products of fusion being then "base bullion," consisting of metallic lead, which contains all the silver and gold present in the furnace charge, and a slag consisting of silicates, usually of iron, lime, alumina, magnesia, &c., according to the nature of the fluxes employed. Obviously auriferous concentrates, consisting, say, of iron and arsenical pyrites, can be employed in this process by being first calcined as sweet as possible. There will thus be obtained an auriferous

oxide of iron which could be added as a flux to the other ingredients of the furnace charge. Practically the whole of the gold present will alloy with the lead produced, and will be found in the base bullion, from which it is afterwards separated by a series of processes. According to the nature of the other ingredients of the furnace charge, it may sometimes be a positive advantage to have a certain amount of crushed quartz left in the concentrates, but as a general rule the aim of the mill man should be to produce as clean and as rich concentrates as possible. In the above process copper may be substituted for lead smelting, a bath of the former instead of the latter metal being used to collect the precious metals. This process is particularly suitable when the concentrates contain a notable proportion of copper pyrites, as the copper would thus become a valuable ingredient of the ore, and be paid for accordingly. This copper process is coming rapidly into favour, coarse copper being produced by a series of smelting processes carried on either in reverberatory or in blast furnaces; the coarse copper is run into slabs, which are then refined electrolytically. During the process of electrolysis the silver and gold, in a very impure state, are deposited as a blackish mud at the bottom of the vats in which the electrolytic refining takes place. This mud is collected and the precious metals obtained from it by cuppellation. This method is being practised in several places in England notably at Swansea, and is applicable to most classes of concentrates; for concentrates carrying a comparatively large proportion of quartz it is perhaps the most suitable. The first operation usually consists in smelting a matte, which will contain all the valuable ingredients of the whole charge. If it were feasible to do this at the mine at a moderate cost, it would be very

advisable to do so, as the matte would be in a better form for shipment than finely-pulverised concentrates; and as a considerable saving in freight would result, seeing that the gold can be concentrated in a matte to a very considerable extent without any risk of loss. There are various smelting works in England, America, and on the continent of Europe, &c., that are prepared to purchase auriferous concentrates in any quantity. This is always done upon assay of a sample, the price given varying according to the richness of the concentrates and their composition, whilst each smelting works regulates the prices offered according to its needs for the time being of such ingredients for making up the furnace charges. No fixed scale of prices can accordingly be quoted. One American smelting works buys at the rate of about $18 for the ounce of fine gold, less a varying charge for treatment. It may be noted that $18 per ounce of gold is equal to 87 per cent. of the full value.

In the Cripple Creek district smelters buy on the same scale as do the chlorinating and cyaniding mills. In England there are firms that buy concentrates for smelting at London, Swansea, St. Helens, Birmingham, and Newcastle. The following table will serve to give a general idea of the prices that concentrates of varying richness will fetch in England, but, as already said, exact figures cannot be given :—

For ore assaying per ton of 2,240 lbs.	Price paid for gold per oz. troy. £ s. d.
5 oz. of gold	3 15 6
10 ,, ,,	3 15 6
13 ,, ,,	3 17 0
16 ,, ,,	3 17 6
20 ,, ,,	3 18 0
30 ,, ,,	3 18 9
40 ,, ,,	3 19 3
50 ,, ,,	3 19 9

For ore assaying per ton of 2,240 lbs.	Price paid for gold per oz. troy. £ s. d.
60 oz. of gold	4 0 0
80 ,, ,,	4 0 9
100 ,, ,,	4 1 0
150 ,, ,,	4 2 3
over 150 ,, ,,	4 2 6

The above table only applies to sulphurets containing but a small proportion—say under 10 per cent.—of zinc. When this metal is present in large quantities, the price is lower in proportion to the amount present. In comparing English and American prices, it must be remembered that the latter are paid in net cash, whilst in the former country, the payment is made in bills at two months.

In West Australia, the charges of the Dry Creek Smelting Works are as follows:—

Assay of ore.	Proportion of gold contents paid for.	Less smelting charge. s. d.
Over 8 oz.	95 per cent.	45 0
From 6 oz. to 8 oz.	94·5 ,,	47 6
,, 4 oz. to 6 oz.	93·5 ,,	50 0
Under 4 oz.	92·5 ,,	50 0

The gold is valued at £4 per oz.

Shipping of Concentrates.—This is almost universally performed in bags; as a general rule, bags made of fine gunny sacking, holding about $\frac{3}{4}$ to $1\frac{1}{2}$ cwt. each, are employed. The material of the bags should be of very close texture so as to prevent any loss of the contents which are frequently in a state of very fine division. It is highly advisable to thoroughly dry the concentrates before bagging them, either by natural or artificial means. Not only will the expense of paying freight upon a certain proportion of water be thus avoided, but, what is even more important, the bags are far more likely to arrive at their destination in good condition. Wet

sulphurets are apt to heat and oxidise, forming sulphates and some sulphuric and sulphurous acids, which rapidly rot and destroy the material of the bags. This is a matter which must not be lost sight of whenever sulphurets have to be shipped for long journeys. It is of course a very secondary matter when the chlorination works are owned by the mine and are consequently close to the mill. For shipment to a distance, however, it might even be advisable to calcine before shipping whenever fuel is cheap at the mine; calcined ore carries better than raw ore, and moreover, in cases where the concentrates consist chiefly of iron pyrites, a saving in freight would result, pure iron pyrites completely calcined into peroxide of iron losing one-third of its weight, so that 3 tons of raw pyrites only carry as much gold as 2 tons of the same calcined, and naturally cost 50 per cent. more for freight. The percentage of moisture present should always be determined when the parcel is weighed before bagging. Before a pile of sulphurets is bagged, the pile should be thoroughly turned over, well mixed, and a fair sample taken out for assay, in order to control the results obtained by the customs works purchasing the parcel.

Cyanidation—This comparatively modern process is based upon the well-known fact of the solubility of gold in a solution of commercial cyanide of potassium to which air has access (see page 28). It does not appear to be equally suitable to all ores, but works well with most, and will extract gold from some on which other processes have failed. As the solution employed has, unlike chlorine, practically no action upon the native sulphides usually occurring in concentrates, calcination is not necessary, except for mechanical reasons in some

cases, and the process is accordingly in so far a cheaper one than chlorination. It has received its principal development in South Africa, and notably on the Witwatersrand gold fields, where it is now worked on a very large scale indeed. Its principal application there is to tailings from which the sulphurets may or may not have been removed by concentration, but both sulphurets and tailings can be and have been treated by it with great success. In 1891 the Robinson mine commenced to erect works for the treatment of its tailings by the cyanide process, and in that year already nearly 44,000 ounces of gold were extracted by it. In 1897 close upon 3,800,000 tons of tailings were treated, yielding 934,545 ounces of gold, worth nearly three millions sterling, whilst the treatment of concentrates by cyanidation produced 57,455 ounces of gold. The principle consists simply in allowing a weak solution of cyanide of potassium, containing usually from $\frac{1}{4}$ to $\frac{1}{2}$ per cent. of commercial cyanide, to percolate through the crushed ore; it is found that such a solution dissolves out a large proportion (in favourable cases over 90 per cent.) of the gold contents of the ore, whilst scarcely attacking any of the base metals that may be present. The solution, which then contains gold in the form of potassic aurocyanide, is filtered off and the gold precipitated from it; as a general rule the exhausted solution still retains some cyanide and is pumped back to be used over again. Sometimes it is allowed to run to waste. The precipitated gold is melted into bullion. Such is briefly the *rationale* of the process of cyanidation; a full discussion of the details of its practical application would be foreign to the object of the present work, and

it will be described only in general terms, so as to supply just as much information about it as may be of use to the mill man.[1]

Plant.—This essentially consists of dissolving tanks in which the solution is prepared, storage tanks in which it is diluted to the desired extent, leaching tanks in which the lixiviation proper is carried out and precipitating tanks in which the gold is deposited. The character and arrangement of the plant will vary with the physical features of the site available, and the quantity and nature of the material it is required to treat. The latter consists, in most cases, of tailings which have been allowed to settle in suitable reservoirs and are thence trammed to the cyanide plant. The experiment of running tailings direct from the mill into the leaching vats has been tried, but presents considerable difficulties, as these tailings settle very solidly, and offer great resistance to the percolation of the solution. At present, direct filling is not much in favour. The first stage, whatever method of filling be adopted, is to get rid of the slimes, whilst a rough classification by means of a few large Spitzkasten or Spitzlutten is often introduced with much advantage. In the most modern plants a system of intermediate filling is used; in this method the pulp is filled direct, often by means of distributors, into a vat, where the sand is allowed to settle and drain, and whence it is transferred to the leaching vat. These intermediate vats are now often supported on iron or brick columns above the leaching vats, an

[1] The reader who desires fuller information may be referred with advantage to Bulletin No. 5, of the California State Mining Bureau, on the cyanide process by Dr. A. Scheidel.

arrangement that reduces the labour of re-handling to a minimum, though it necessitates the lifting of the mill tailings to a considerable height.

Dissolving Tank.—This is sometimes dispensed with, the solid cyanide being then placed beneath a stream of water flowing into the leaching tank. Usually it is a small tank or vat, into which the crude cyanide is thrown, water being run in until it is dissolved. The strength of the solution is then determined, and it is run off through a filter of fine steel wire into the storage tanks, where it is diluted as required. The insoluble impurities present in the crude cyanide settle in this small tank, which has to be cleaned out from time to time. Crude cyanide is shipped in sealed canisters containing 190 to 195 lbs. of the cyanide, which usually contains between 75 and 98 per cent. of pure potassic cyanide; the remainder is generally carbonate and cyanate of potash.

Storage Tanks are precisely similar in structure to the leaching tanks about to be described, except that they contain no filter.

Leaching Tanks.—There are several types of these, cement or brick tanks and wooden and steel vats; the first-named are considered out of date, whilst the use of steel is apparently thought very well of. Cement tanks are at times, as at the Langlaagte Estate, sunk below the surface of the ground and tram-lines run over and alongside them for charging and discharging them. The latter process is effected by means of a steam-crane which hoists a grab capable of filling two trucks at each lift. By these means a tank can be emptied in 14 hours at a cost of 2*d*. per ton. The cyanide solution has a disintegrating action upon cement, and the insides of these

tanks should be painted with a thick coat of paraffin paint; this precaution is especially necessary with tanks excavated in the ground, leakage from which is apt to escape detection. Large vats with bottom discharge are now considered the best, and for this purpose, brick and cement are unsuitable; wood or iron vats carried on brick piers or iron columns with lines of rail running below the vats have been used in the most modern plants. Wooden and steel vats are generally preferred; they have been made up to 40 feet in diameter, according to the size of the plant:—

A vat	14	feet in diameter,	5	feet high,	holds about	50	tons of	tailings.
,,	22	,, ,,	5	,,	,,	100	,,	,,
,,	22	,, ,,	6	,,	,,	130	,,	,,
,,	40	,, ,,	8	,,	,,	360	,,	,,

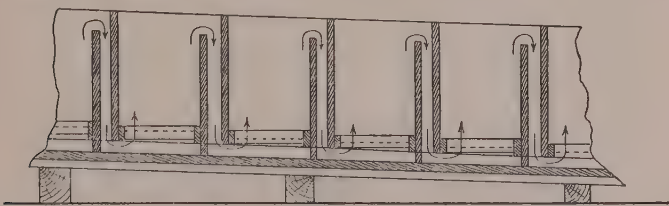
The above quoted dimensions are all from actual practice, the last-named exceptionally large vats having been erected at Roodepoort. Wooden vats should be made of staves 3 inches to 4 inches thick, of thoroughly well-seasoned lumber, held in place by hoops of bar-iron, the ends of which are drawn together by powerful bolts. The bottom should be made of good sound 4-inch plank let into the staves. All the work should be most carefully done. The joints should be made with thin white-lead, and when finished the vat should be thickly coated with paraffin paint. The filter consists of a porous false bottom a few inches above the actual bottom of the vat; it is sometimes carried on a wooden grating which rests on a bed of clean gravel, but more often on a frame of stout wooden slats a couple of inches deep, or of tiles pierced with 1-inch round holes. This filter frame is about 1 inch less in diameter than the vat itself; upon it is laid the filter proper, made of very

stout blanketing, or, better still, cocoanut-matting, held in place by a gasket of rope driven well down all round it. Grooves are cut in the bottom of the vat leading towards the mouth of the delivery pipe, which generally consists of a piece of very stout india-rubber hose pipe. For iron or steel vats the side plates are best made $\frac{3}{16}$ inch to $\frac{5}{16}$ inch thick, with vertical joints only, lap-jointed, riveted, and caulked; the bottom plates should be about $\frac{1}{8}$ inch thicker than the sides. The sides should be stiffened by one or more rings of angle iron riveted round them. The vats should be painted inside and outside with asphalt paint.

The exhausted tailings are mostly discharged through side gates or through bottom doors of various kinds. They are best made of steel-plate, and are held water-tight by means of bolts against india-rubber seatings. Whenever possible the exhausted tailings are sluiced out of the vats by means of a head of water. When this is not available they are emptied by hand; this takes from six to nine hours, and is not a very expensive matter in the Transvaal, where Kafir "boys" are employed for such duties. It may be mentioned that rectangular wooden tanks, though cheaper to construct than round ones, are not found to work so well in practice.

Precipitating Tanks, or "zinc boxes" as they are also called, are used for collecting the gold. The precipitant hitherto mostly employed is zinc shavings. These are prepared by threading a number of thin discs of zinc on a spindle and turning the cylinder so formed in a lathe. One man can produce about 50 lbs. of zinc shavings in a day's work. These shavings are carried on trays about 20 inches square, the bottoms of which consist of iron wire-gauze of $\frac{1}{4}$-inch mesh, the trays resting on cleats

about one-fourth way up in the box. A rule sometimes followed is to allow 2 cubic feet of zinc shavings for every ton of solution that passes through the box in 24 hours. The precipitating boxes are usually about 20 to 30 feet long, 2 to 4 feet wide, and 2 to 3 feet high. They are divided into about ten compartments by transverse partitions; the first one of these does not reach to the top of the box, the next one not quite to the bottom, and so on, so that the solution is compelled to flow alternately up and down in its passage through the box, as shown in the diagrammatic section, Fig, 105. The first



VERTICAL SECTION
Scale, $\frac{1}{3}$ inch = 1 foot.
FIG. 105.

compartment is left empty to act as a settling-box for any fine sand that may be carried over by the solution, and the last two are similarly left empty for fine gold to settle in, that might otherwise be carried away by the solution. Each tray holds about 40 lbs. of shavings. There are usually at least two sets of precipitating boxes.

From the precipitating boxes the solution runs into wooden vats or cement tanks, as the case may be, to be pumped back as required. All pumps, cocks, valves, &c., about a cyanidation plant should be made of iron, as brass is slightly attacked by the solution.

The Leaching Process.—The tailings are charged, best by hand, into the leaching tanks until these are filled to within about 6 inches from the top. As a rule, tailings, unless very recent, contain soluble salts, owing to the oxidation of more or less of the sulphurets originally present; these salts have to be washed out. The leaching tank is supplied with a water tap, which is turned on and water run in till the tank is full. The water is then allowed to stand until these salts are completely dissolved, a matter which has to be ascertained in every case by experience; the water is then run off. If, as is mostly the case, the production of these salts has been accompanied by that of free acid, the latter has to be neutralised; this is done by running in a solution either of caustic soda or, better still, of lime-water. This is then run off, and then the cyanide solution is run in. The strength of this is generally between 0·2 and 0·5 per cent. (4 to 10 lbs. to the ton), but the most suitable strength has always to be determined for each ore by experiment. This solution is allowed to remain for some hours and is then drawn off, being replaced by fresh; this treatment with so-called "strong solution" lasts from eight to twelve hours. After the "strong solution" leaching is completed, "weak solution," containing from 0·08 to 0·2 per cent. of cyanide, is run in, This "weak solution" consists generally of the first solution after the gold has been precipitated out of it, made up, if necessary, to the proper strength. This is allowed to act for another eight or ten hours, and is then run off. The "strong solution" and the "weak solution" are run through separate zinc boxes. The bullion obtained from the former is finer than that obtained from the latter; the solution from the former is

caught in tanks, that from the latter is often run to waste. After the action of the solution is complete, a quantity of wash-water is run into the vat about equal to that which ultimately remains adherent to the tailings, so that the volume of the solution remains undiminished. About half a ton of "strong" and half a ton of "weak" solution are used for each ton of ore in the vat, and are thus kept in circulation. The treatment of each vat takes about three days. Experiments have been tried on a system of continuous circulation by which the same solution has been made to pass through several leaching vats, or several times through the same one, but the advantage of this method is by no means established as yet. It is, however, stated to have been worked with success at the Gold Run Mine, Siskiyou County, California.[1] Here a small four-stamp mill crushes 5 tons of quartz every 24 hours, inside amalgamation being employed, which collects in the mortar 86 to 90 per cent. of the gold saved. The pulp passes over the usual amalgamated copper-table, and thence direct into a "pulp tank" 4 feet deep and 6 feet in diameter. The overflow from the pulp tank, whilst the latter is being filled, runs into a "sump tank," where the slimes settle, to be drawn off as they accumulate, into a "slime tank" for treatment. The water from the sump tank is pumped back into the battery, being used continuously. There are four pulp tanks, each of which is fitted with a false bottom carrying a filter; when one is full, the stream of pulp is turned into the next one. The pulp is then drained as dry as possible, and cyanide solution containing $\frac{1}{3}$ to $\frac{1}{2}$ per cent. of cyanide is pumped in. Thence it runs through the

[1] *Eleventh Report of the State Mineralogist, California State Mining Bureau*, page 430.

zinc box, which contains twelve compartments, 8 inches square, and, after the gold has been precipitated there, into two cyanide solution tanks, where it is brought up again to its full strength, and is then ready for another round. Usually each tank of pulp is treated four times. It is said that this process leaves a mere trace of gold in the exhausted tailings. The total cost of milling and continuous cyanidation is said to be \$6 per ton, whilst the gold saved amounts to between \$50 and \$120 per ton by amalgamation, and \$10 to \$33 by cyanidation. Continuous cyanidation is being tried in Australasia; it does not seem to have been found successful in South Africa.

Precipitation.—The solution is run through the zinc boxes already described, each box treating about thirty tons of solution in nine hours. The zinc shavings in the first trays are dissolved away more rapidly than in the last ones; the shavings from these are therefore used to fill up the front trays, fresh bright shavings being used to refill the last ones. The solution, when it runs into the zinc boxes, carries usually from 1 to 3 ounces of gold to the ton, and should not contain more than 12 grains when it escapes. In a few places zinc fume (finely divided zinc obtained in zinc smelting) is employed as a precipitant.

Collecting the Gold.—This is done about once a fortnight; the trays are lifted out and washed, and the slimes of impure gold are allowed to settle. The clear water is syphoned off, and the slimes scraped out and rubbed through a fine screen into the small tank, the pieces of zinc remaining on this screen being returned to the trays on top of the shaving. The impure gold is again allowed to settle, and transferred to enamelled iron pans, in

which it is dried. It is sometimes simply mixed with borax, sand, and carbonate of soda, and melted in clay crucibles, the ordinary melting furnace, to be subsequently described, being used for this work; it should, however, be furnished with dust chambers so as to collect as much as possible of the zinc fume formed in this operation, which always carries gold. More often the dried slimes are first calcined in reverberatory or muffle furnaces, the bed of which consists of a shallow iron pan. In South Africa the slimes are usually heated with nitre until most of the base metals present are oxidised. The bullion produced is very impure, hard, brittle, and greyish in colour; it is about 650 per 1000 fine. Besides zinc, it contains, silver, copper, and small quantities of other metals. It is usually shipped without further refining.

Another method, preferred in America, is to treat the gold slimes with dilute sulphuric acid in wooden tubs, by which means a large part of the zinc present is dissolved. The remaining slimes are washed by decantation, dried in a filter press, and calcined in a muffle at a low red heat, and the residue then fused as above. It is said that in this way bullion over 900 per 1000 fine may be produced.

Whatever method be employed the slags always contain a notable proportion of gold; some of this can be recovered by crushing, panning, and amalgamating, but some of the gold resists this treatment and can only be extracted by smelting with lead.

Theory of the Process.—This is still somewhat imperfectly understood. The equation already given (page 28), which purports to show the actual chemical reaction that takes place when gold is dissolved in cyanide solution, is

by no means the only reaction which occurs. A solution of potassic cyanide, even when pure, undergoes numerous decompositions, chiefly through the action of atmospheric oxygen, cyanates, formates, carbonates, ammonia, and various other compounds being formed, all of which no doubt influence, even if they do not take an active part in the reaction; the consumption of cyanide is accordingly enormously in excess of that required by the above equation. In practice it amounts to more than 3 lbs. of cyanide for each ounce of gold precipitated, or from 0·4 to 0·8 lb. per ton of tailings treated.

The reactions involved in the precipitation are also decidedly obscure; thus it is still doubtful whether pure zinc would precipitate gold from a solution of pure potassic aurocyanide. What occurs in effect is that zinc goes into solution as a double cyanide (K_2ZnCy_4), and that hydrogen is evolved. It is probable that the free alkali, which seems to be always present, attacks the zinc, forming a series of electric couples with the other metals present, and that under the influence of this electric current double decomposition ensues between the zinc and the potassic aurocyanide. The whole subject still needs much investigation, and it is not yet even fully known what are the products generated during the action. The practical result is that gold is precipitated, and is not redissolved as long as there is any metallic zinc present. The consumption of zinc is much greater than is indicated by the chemical reactions generally supposed to take place, being in practice between $\frac{1}{3}$ lb. and 1 lb. of zinc to the ounce of gold precipitated. It is important to remember that none of the cyanogen present in the above zinc compound is available for dissolving gold again.

Cost of Cyanidation.—The Transvaal is responsible for

rather more than two-thirds of the total amount of gold extracted in the whole world by cyanidation; it is therefore to this country that we have to look chiefly for reliable data as to the cost of the operation. This is between 2*s*. and 4*s*. per ton treated on the Witwatersrand, but may rise to between 7*s*. and 15*s*. in outlying districts such as De Kaap and Lydenburg. The average cost of charging and discharging vats is about 10*d*. per ton, and of general labour about 6*d*. per ton, whilst power, water, maintenance, &c., are very variable. The consumption of cyanide is usually from 0·3 to 1 lb. per ton, costing say 4d. to 1*s*. per ton, whilst the consumption of zinc is about 0·25 lb. per ton. The following table shows the detailed costs per short ton for a few typical Witwatersrand mines that are worked on a large scale:—

	I.	II.	III	IV.	V.
Tons treated per annum.	88 928.	139,076.	180,980	127,763.	13,650.
	s. *d.*	*s.* *d.*	*s.* *d.*	*s.* *d.*	*s.* *d.*
European wages..	0 5·30	0 4·16		0 4·22	0 2·40
Native wages, including food ...	0 4·44	0 7 28	1 4 78	0 1·40	0 1·37
Cyanide...........	1 1·20			0 9·66	1 7·15
Lime	0 0·92		0 4·57	0 0·40	0 0·68
Zinc	0 1·20	1 4·12		0 1·79	0 3·80
Fuel (power)......	0 1·18		0 5·16	0 0·75	0 1·10
General stores ..	0 2·96			0 2·17	0 0·78
Maintenance......	0 3·66	0 6·36	0 7·67	—	—
Miscellaneous ...	0 0·70	0 1·56		0 0·55	—
Contractors	0 1 45	—	—	0 10·46	0 8·46
Total	2 11·01	2 11·48	2 10·18	2 7·40	3 1·74

I. Jumpers Gold Mining Company, Limited. II. Crown Reef Gold Mining Company, Limited. III. Langlaagte Estate and Gold Mining Company, Limited. IV. City and Suburban Gold Mining Company, Limited, Tailings. V. Ditto, Concentrates.

The cost of treating concentrates by cyanidation is usually much higher than that of treating tailings, on account of the much greater consumption of cyanide, which may become practically prohibitive if the concentrates contain sulphides of copper or of other base metals readily attacked by cyanide solution. The cost of treating tailings at the Langlaagte Estate works is as shown above (No. III.) 2*s*. 10·18*d*. per ton, the consumption of cyanide and of zinc being 0·323 lb. and 0·177 lb. respectively; the cost of treating concentrates at the same works mounts to 12*s*. 1·91*d*., the consumption of cyanide being 5·40 lbs. and of zinc 4·46 lbs.

In America cyanidation has been used chiefly at the Mercur and other mines in the Camp Floyd district, Utah, and at Cripple Creek, Colorado; in the former the cost per ton, including that of crushing the ore, was as follows during 1893:—

	$ cents.
Cyanide (1·27 lbs.)	0.66
Zinc (0·55 lb.)	0.05
Supplies, fuel, repairs, &c. . . .	0.57
Labour	1.12
Total	2.40 = 10*s*. per ton.

This figure has since then been considerably reduced. In the Cripple Creek district the cost is said to exceed $4 (16*s*. 8*d*.) per ton, but this figure appears to include the cost of crushing the ore as well as of calcining it, better results being obtained with calcined than with raw ore. It is said that this improvement is due to physical changes merely; the Cripple Creek ores contain tellurides, but it is not known whether these compounds have any effect upon the leaching process. A cyanide plant has been in operation for some time at the property of the Standard Consolidated Mining Company, Bodie, Mono County,

California treating about 80 tons per day, which are hauled a distance of 1,400 feet up an incline by electrical power, a 10 H.P. motor being used, together with a 5 H.P. motor to drive the pumps, &c. The cost of operating the plant is as follows:—

	s.	d.
Labour	2	3·110
Cyanide (0·42 lb.)	1	0·580
Zinc (0·29 lb.)	0	1·280
Lime (7·30 lbs.)	0	5·475
Sulphuric acid (0·13 lb.)	0	0·250
Wood	0	5·230
General supplies	0	1·155
Miscellaneous	0	0·255
Haulage	1	3·315
Total	5	8·650

A few other cyanide plants are in operation in California and in one or two of the other States, but the process has made comparatively little headway in America.

In Australasia, cyanidation is being adopted in several of the West Australian mines. In New Zealand it has been rather largely employed, the average cost there being about 4*s*. to 5*s*. per ton. At Hannan's Brownhill Gold Mining Company, Limited, West Australia, the cost has recently been 10*s*. 9·4*d*. with a monthly treatment of 1,040 tons. At the Lake View Consols, the cost of cyanidation by leaching is given as 7*s*. 3*d*. per ton, and by filter press as 13*s*. 5*d*; at the mill of the Ivanhoe Corporation these costs were as low as 5*s*. 3*d*. and 6*s*. 3*d*. respectively; the cost of cyanidation at the Great Boulder Proprietary is stated to have been 11*s*. 9*d*. At the Victory Gold Mining Company, Limited, Charters Towers, Queensland, the cost of cyaniding 4,406 tons of tailings amounted to 16*s*. 9*d*. per ton, out of which 7*s*. 2*d*. per ton was paid in wages.

In the Mysore district of India cyanidation has proved

very successful; the following are the costs of three of the leading companies :—

	Tonnage treated per annum.	Cost per ton s.	d.
Mysore Gold Mining Company, Ltd.	20,015	4	0·9
Champion Reef Gold Mining Company of India, Limited . . .	73,302	4	7·1
Nundydroog Company, Limited . .	25 931	2	6·9

The first cost of a cyanide plant varies very widely, according to local conditions and the character of the plant itself. Broadly speaking, it may cost from £1 for each ton to be treated monthly up to three times that amount.

Siemens and Halske Process.—In this process the method of solution is the same as in the original one, but electrolytic deposition is substituted for precipitation by zinc. The latter is admittedly the weakest point of the MacArthur-Forrest process, and it seems to fail altogether when applied to dilute solutions, which latter can be treated successfully by the electric current. Therefore in the Siemens and Halske process far more dilute cyanide solutions are used, the time of treatment being correspondingly extended; the strong solutions contain 0·05 to 0·1 per cent. of cyanide, and the weak solutions 0·01 to 0·05 per cent. the time of treatment being between four and ten days. The precipitating boxes are wooden tanks 20 to 30 feet long, 6 to 8 feet wide, and about 4 feet high, in which the electrodes are suspended, these being so arranged as to cause the solution to travel up and down in the same manner as in the zinc boxes. The cathodes consist of thin strips or sheets of lead carried on light frames, and the anodes of sheets of iron about $\frac{1}{8}$ inch in thickness, the latter being enclosed in canvas, so as to retain the Prussian blue formed during the electrolysis of the

solution. Only a weak current is required, about 0·06 ampère per square foot being sufficient; with cathodes $1\frac{1}{2}$ to 2 inches apart a 5 to 7 volt current is ample. A box capable of treating 3,000 tons of tailings per month, or about 100 tons of cyanide solution per day, requires about 10,000 square feet of cathode surface, and will need 5 H.P. to generate the requisite current. Under the action of such a current, the gold is deposited in a coherent film on the sheet lead; when the film has reached the desired thickness, the frame carrying the lead is withdrawn, the gold-bearing lead removed and melted up, forming base bullion that may contain up to 12 per cent. of gold. The base bullion is treated by Parkes's process, which consists in melting it with zinc, when an alloy of zinc, gold, and some lead is produced, the bulk of the lead being entirely freed from gold. The zinc is then distilled off from the auriferous alloy, and the resulting enriched lead is cupelled, the bullion thus obtained being about 900 per 1,000 fine. This process, whilst still to a certain extent on its trial, is already producing no inconsiderable quantity of gold; out of the total amount of gold stated above (page 408) as having been extracted in 1897 from tailings by cyanidation, over 150,000 ozs., or about one-sixth of the whole, were precipitated by the electrolytic method.

This process has hitherto been almost confined to the Transvaal, hence it is only there that its costs have been determined with any accuracy. The following figures for 1896, taken from the Proceedings of the Chemical and Metallurgical Society of South Africa, Vol. I., show (I) the estimated cost of treatment per ton in a plant working 500 tons of tailings per day, and (II) the actual cost of three months' operations at the May Consolidated Gold Mining Company, Limited,

during which time 27,250 tons of tailings were treated, producing about 5 dwts. of bullion to the ton :—

	I.			II.				
	Costs per ton.			Actual costs.			Costs per ton.	
		s.	d.	£	s.	d.	s.	d.
Filling and discharging		0	10·0	1,197	19	2	0	10·5
Cyanide	(¼ lb. at 1s. 1½d. per lb.)	0	3·4	284	0	0	0	2·5
Lime and caustic soda		0	1·7	—			—	
Lead	(¼ lb.)	0	1·1	105	0	0	0	0·9
Iron	(⅓ lb.)	0	2·2	434	9	10	0	3·8
Stores and general charges		0	3·2					
Assaying and refining charges				159	5	0	0	1·4
Fuel and power		0	4·0	248	12	3	0	2·2
White labour		0	5·0	401	5	0	0	3·5
Native wages and food		0	1·9	91	8	0	0	0·8
Maintenance		—		166	4	9	0	1·5
		2	8·5	3,088	4	0	2	3·1

The above figures are exclusive of interest on plant and royalty. The May Consolidated plant cost £27,500, so that interest is a heavy item, whilst royalty amounted to about 4·5*d.* per ton. Upon the whole, it may be said that the cost of treating tailings by the Siemens-Halske process on the Witwatersrand is about the same as by the ordinary cyanidation process, although the loss of cyanide is decidedly less; it seems to give a somewhat lower percentage of extraction, and the first cost of the plant is decidedly greater. On the other hand, it produces much finer bullion.

Slimes Treatment.—This forms the most recent development of the cyanidation process. The slimes are either deposited in large settling pits, and thence transferred

to the vats, or in the more modern plants are run direct into leaching vats or first into intermediate vats. The leaching vats are provided with agitators, consisting of vertical shafts, provided with arms, or else the pulp in them is agitated by being kept in circulation by centrifugal pumps or by injecting air, any of these methods appearing to give satisfactory results. The slimes are treated in the vats with three to four times their weight of a very dilute cyanide solution, containing 0·008 per cent. of cyanide; after about two hours' agitation the slimes are allowed to settle, and the clear solution is run off into settling tanks, and thence passes to Siemens and Halske electrolysis boxes; the zinc precipitation method cannot be used on these very dilute solutions. The exhausted tailings are now simply discharged, attempts at washing out the absorbed cyanide solution having so far proved unprofitable. A charge of about 60 tons is treated in 12 to 18 hours, and an extraction of 75 to 80 per cent. of the assay value of the slimes is realised.

The costs of slime treatment seem to range between 3*s*. 6*d*. and 4*s*. 6*d*. per ton treated. The following table shows the cost at the Crown Reef Gold Mining Company, Limited, for the year ending March 31st, 1898, during which 40,955 tons of slimes (22·12 per cent. of the tonnage milled) were treated:—

	s.	*d*
European wages	0	8·23
Native ,,	0	3·40
Stores and materials	1	3·52
Maintenance	0	10·26
Electric pump	0	3·98
,, light	0	0·46
Royalty	0	2·27
Total	3	8·12

At the Geldenhuis Estate a slimes plant capable of treating 3,000 tons a month has been erected at a cost of £24,000; the cost of treatment there was 6*s.* 5*d.* per ton for the first four months of its operation.

First in the Transvaal and more recently in Western Australia slime treatment has become a separate branch of the art of gold extraction; in the latter place the filter press has been successfully employed for freeing the treated slimes from adhering cyanide solution; as has been already indicated, this adds somewhat to the cost of treatment, but renders it possible to treat slimes that would otherwise be practically intractable.

The introduction of the cyanide process has had a very marked effect upon the whole system of gold extraction, and it may now be looked upon as the recognised means of treating tailings. In spite of the highly poisonous character of the solution, fatal accidents are all but unknown; some men seem especially susceptible to its effects, and in many cases contact with the solution produces painful sores on the arms and hands of men engaged in working with it; there is, however, little need to touch the solution, except during the clean-up. This operation, and indeed the whole of the zinc precipitation, would appear to be the weakest part of the entire process. Hitherto, however, nothing has been devised that has succeeded in taking its place. The Siemens and Halske process is still on its trial.

Molloy's process was a very promising one, but has not come into use; it consists in replacing zinc by sodium, which is employed in the form of an amalgam, the latter being produced electrolytically and simultaneously with its consumption by an ingenious piece of apparatus. Moreover, in this process sodic cyanide, which is quite

as effective a solvent of gold as potassic cyanide, is regenerated by the precipitation of the gold. Nevertheless, the process has not been a success practically. Several inventors have attempted to devise an electrolytic method, using mercury or some amalgam as an electrode, but hitherto without success.

For ordinary battering sands or crushed ores the simple percolation method is still considered the best, and it is upon the whole surprising how very minute a percentage of the vast number of patents annually applied to cyanidation processes, can survive even the experimental stage. The process does not seem to be equally applicable to all ores, although it is difficult to see under what conditions the gold would refuse to dissolve; obviously the physical character of the gold is an important element in the problem. Cyanidation has been found applicable to many ores, and especially to some that carried no freely amalgamable gold. Such ores can be treated by crushing dry, and cyaniding the crushed ore. If desired the exhausted tailings could then be sluiced out of the vats and over amalgamated plates or blankets to catch any coarse gold that might be present. Such methods are in use in several places; the great obstacle to their more extended introduction lies in the fact that there does not exist at present any satisfactory dry crusher that can deal with hard material at a cost at all comparable with that of wet stamping.

CHAPTER XIII

CLEANING-UP—TREATMENT OF AMALGAM—CLEANING, RETORTING, AND MELTING

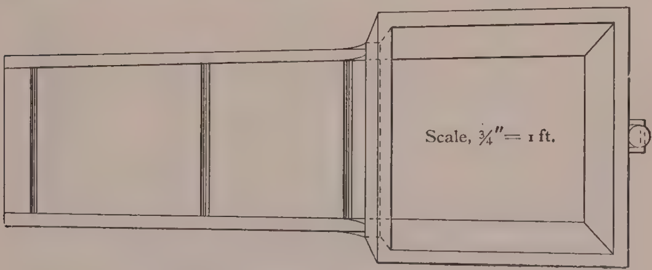
Battery Clean-up.—This takes place weekly, fortnightly or monthly. As a general rule, inside plates are cleaned-up every week, but a complete clean-up of the whole mill is made only once a month. To effect this, the ore-bin gate is closed, and the ore-feeder wheeled back out of the way; the battery is allowed to run until all the stamps are pounding upon metal, only ten heads being thus treated at a time. If the mill is provided with suitable gear, it is advisable to "beat down" at a slower speed than the normal rate of milling, say at 40 to 50 drops per minute. The stamps are then hung up, the water being allowed to continue running until it comes away quite clear; then it too, is stopped. The outside plates are then thoroughly cleaned and scraped, as already described. The upper portions of the table are next covered over with a stout wooden cover, or with pieces of plank, which reach right across the two tables if ten heads are being cleaned at a time. These serve for the workman to stand on whilst cleaning out the mortars, and to protect the plates from injury. The screens are next unkeyed and laid aside, and the insids plates and chock-blocks are taken out and care-

fully scraped and cleaned, the plate being usually unscrewed from its seat for this purpose. As much sand and coarse ore as can be got out of the mortar by means of short-handled scoops and trowels is then scraped out, and piled either on a clean spot on the mill floor or in a bin kept for the purpose. The middle die of the mill is next prised out. If the mortar is fitted with any of the arrangements already described to facilitate this, it becomes an easy matter; if not, it usually involves a good deal of work with steel bars and sledge-hammers, cold chisels, and wedges, before the die can be lifted. A few bars with pointed and chisel-shaped ends should always be kept in the mill for this special purpose. Once the middle die is out, the others are readily got out by means of a bar. Any sand, &c., adhering to the dies is scraped and washed off, and added to that already obtained. The dies are carefully examined, and any amalgam adhering to them, or collected in cavities in them, carefully removed and put aside. The whole of the sand contained in the box is then dug out by means of scoops, trowels, and amalgam knives, and the liner plates are taken out and treated like the dies. Finally, the inside of the mortar box itself is well cleaned and washed out, and all amalgam adhering to it scraped off and put aside. The shoes are next examined and similarly scraped; especial attention should be given to the groove formed where the head meets the shoe, as this is a favourite spot for the accumulation of amalgam. Any worn-out shoes are next knocked out and replaced by new ones. The liner plates of the mortar are now put back into their places, or, if worn out, replaced by new ones; the dies are put into place upon the false bottoms or upon a bed of tailings well rammed down, the interspaces

between them being filled with small ore, the chock-block and screen are keyed into their places, the outside plates are rubbed up, and the battery is re-started. By this time the next ten heads will have been beaten down, and they are cleaned-up in the same way, and so on until the clean-up of the whole mill is completed. Of course, if there is a sufficiently large staff, two or more ten-head batteries may be cleaned up simultaneously. It is customary to so arrange the work, whenever possible, that the double shift (night as well as day men) shall be available for a clean-up, and a few outside labourers are also sometimes detailed to assist. It need hardly be said that a clean-up offers specially favourable opportunities for theft, so that constant vigilance is demanded on the part of the mill manager, who should really never leave the mill from the commencement to the end of the operation. The contents of all the boxes have now been collected, and are ready for treatment. Here again, if a large staff is available, the treatment of these may commence as soon as the first mortar box is emptied out, and may proceed simultaneously with the cleaning out of the remaining boxes. There are various ways of treating these sands, each of which has its advocates. In a small mill they are sometimes simply panned up; the first panning is done in large prospecting pans in a tub or tank of convenient height filled with water, in which the tailings collect. The final panning of the amalgam is performed in pans, the bottom of which consists of a sheet of amalgamated copper, a little mercury being poured into the pan. This greatly facilitates the cleaning-up and collecting of the amalgam, which, being softened by the mercury added in the pan, adheres to the bottom. Any pieces of iron met with in

panning are put on one side for treatment as described further on.

Sometimes these sands are cradled in a common cradle. When this is done the cradle should be fitted with good transverse riffles $\frac{3}{4}$ inch deep, behind which a little mercury is poured to soften and collect the amalgam. A suitable cradle for this work is shown in Figs. 106 to 108. It is of the usual type, only heavier and stronger, and very carefully made. The hopper is a little deeper than usual, and its bottom is made of a piece of $\frac{1}{8}$-inch

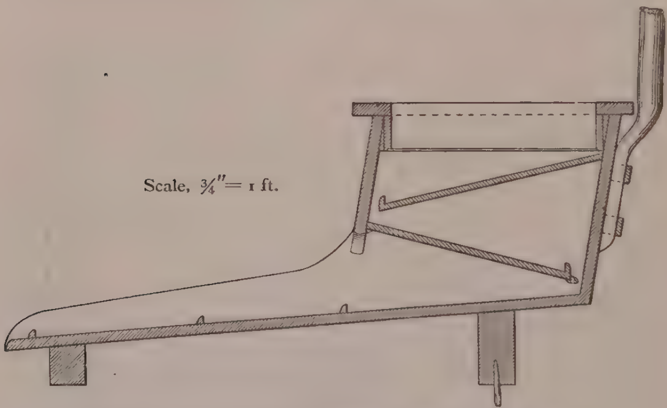


Plan, without hopper.

FIG. 106.

sheet-iron punched with round holes $\frac{1}{2}$ inch to $\frac{3}{4}$ inch in diameter. Stout sills should be laid down for the rockers to work upon, the upper one having a slot into which fits the pin shown projecting from the upper rocker, so as to keep the cradle in its place. A water-pipe $\frac{1}{2}$ inch in diameter, fitted with a valve, and terminating in a couple of feet of hose-pipe, should be arranged so as to supply water to the hopper. The cradle is mostly worked by hand, but may be driven from any convenient portion of the mill shafting. The lower end of the cradle should deliver into a sluice some 10 feet

long, the upper portion of which is lined with an amalgamated copper-plate and the lower portion with riffles. This sluice should deliver into a small pit. A still better arrangement exactly resembles the above in all particulars, except that the cradle, which is then about 10 feet long, is fixed, and cannot rock, being firmly bolted to the sills; the sheet-iron hopper bottom may also with advantage be of somewhat stouter iron. In either case the hopper

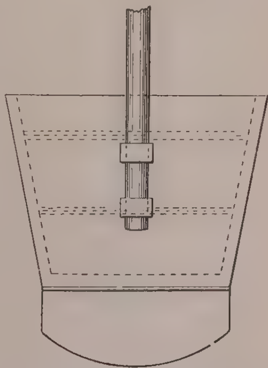


Longitudinal Vertical Section through centre, with hopper.

Fig. 107.

is about one-third filled with battery sands, a sufficient stream of water is turned on, and either the cradle set in vigorous motion in the former arrangement, or the sands rubbed up and down and worked about on the hopper bottom by means of a short stick in the latter one. All the hard lumps, consisting of sand, clay, fine ore, and amalgam, are thus disintegrated and washed down through the riffles, &c., which retain practically all the amalgam, any that escapes being caught in the sluice.

The contents of the hopper are then examined, any large lumps of amalgam (or of more or less completely amalgamated gold) picked out and collected, all lumps of iron thrown together on one side, and the clean stones remaining in the hopper thrown out, to be at once returned to the battery. A fresh lot of sand is then thrown into the hopper, and so on. Ultimately the bulk of the amalgam will be caught in the riffles, and a little may be cleaned up out of the sluice. All the tailings from this clean-up will be caught in the pit, and may be returned to the mill, or else treated separately in the amalgamating barrel, to be presently described.



Back View, without hopper.

Scale, ¾″ = 1 ft.

FIG. 108.

Instead of this method, the battery sands may be treated at once direct in the amalgamating barrel, and this is perhaps the best plan in large mills. In this case all the battery sands, together with a sufficiency of water and a few pounds of mercury, are worked in the barrel for a couple of hours until the amalgam is completely separated from the sands. The Californian amalgamating pan described in Chapter XII. is also occasionally used for this purpose, but is less suitable owing to the fact that big lumps of iron, which would keep the muller from grinding properly, often find their way into the sands. Whichever of the above processes is adopted, the prospecting pan has to be used ultimately to clean-up and collect the now pasty or semi-fluid amalgam, and

especially to separate it from the smaller particles of iron which adhere obstinately to it. This separation is usually accomplished by means of the magnet. The iron in question is derived in part from the wear of the shoes and dies, and in part from fragments of various pieces of machinery, from bolt ends, nuts, pick points, spikes, &c., that find their way into the mortar box with the ore. The ultimate result of the battery clean-up is accordingly to produce :—

1. Tailings, varying in size from sand to large lumps of ore, and which are returned to the battery;

2. Pieces of iron to which more or less amalgam adheres;

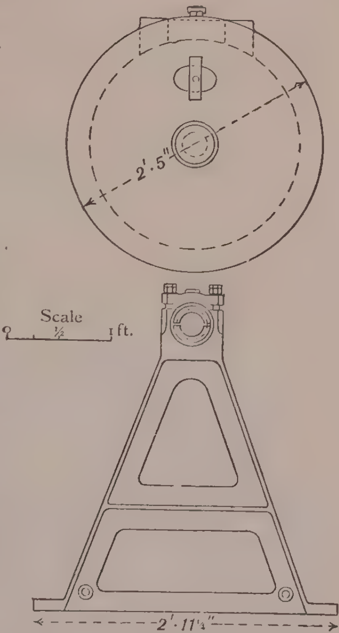
3. Pasty amalgam reserved for further treatment.

With respect to the iron, it may be noted that this is apt to become slightly amalgamated in spots by the use of sodium amalgam, whilst amalgam may also lodge in any crevices in it. All the iron that is collected, is thrown together in a heap on any smooth, level piece of ground near the mill, and there allowed to rust, its oxidation being accelerated by moistening from time to time with a strong solution of sal-ammoniac. When completely rusted through, it is treated with other mill rubbish in the barrel, and any amalgam it may contain recovered.

Amalgamating Barrel.—In addition to the uses already described in cleaning-up, the amalgamating barrel is a most useful adjunct to the stamp-mill for general gold-saving purposes. This barrel is shown in Figs. 109 and 110. It consists of a stout cast-iron barrel, supported on short shafts, which are carried in bearings on a light iron frame. One of these shafts is fitted with fast and loose pulleys by means of which the barrel is caused to revolve.

The barrel has a man-hole large enough to allow of its being rapidly discharged, and has one, or sometimes two, hand-holes in the ends; all these holes are fitted with water-tight covers. A hose is provided which can supply water to the interior of the barrel through a hand-hole, and there is also a movable sluice by means of which the contents of the barrel can be run off.

In all mills there is always being produced a certain quantity of material that carries mercury and amalgam, which latter would be lost unless this material were specially treated for the recovery of any gold it may contain. Such material is, for instance, found in the contents of the amalgam traps. Again, amalgam and mercury are apt to be spilt or to leak from time to time on to the floor of the millhouse; and there is nearly always a certain amount of splashing of pulp from the battery on to the floor. This floor should accordingly be watertight, made either of stout planks, well caulked, or of concrete, and should be so arranged as to slope gently from all sides towards a small pit at one end. Once a



End Elevation

FIG. 109.

day the floor should be washed down by means of a hose-pipe, and cleaned down with a squeegee, everything being thus collected in the pit, where it is allowed to settle, the water being run off, and the contents of the pit, from time to time, shovelled out. All chips of wood taken out of the mortar boxes, and more especially portions of the wedges used for wedging the stamp shoes into the heads, old chock-blocks, screen frames, &c., should be

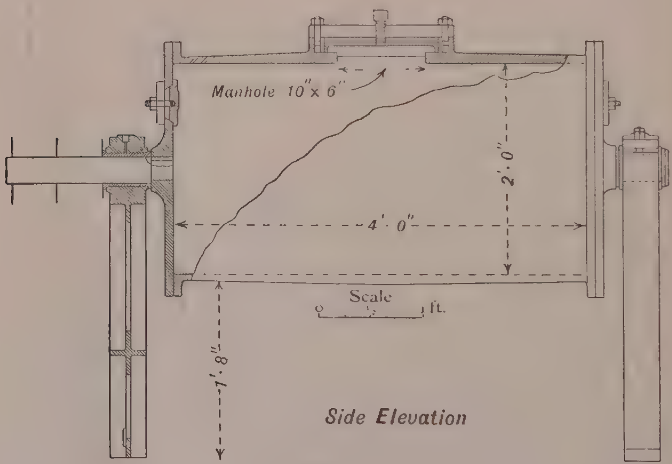


FIG. 110.

collected and dried. Every few months a heap of these fragments should be burnt on a hard, smooth floor, and the ashes, which will be found to contain some gold, collected. All this miscellaneous material is worked up, together with any other rubbish supposed to contain gold, in the amalgamating barrel. It is charged into it through the man-hole in lots of a few hundredweight at a time, the barrel three-fourths filled with water, and a

number of heavy cast-iron balls dropped in. A few pounds of mercury, proportioned to the supposed gold contents of the charge, are poured into the barrel, which is then closed and allowed to revolve slowly, say 20 to 30 times per minute. The mercury used should be purified and charged with sodium, like that used in the mortar box; a little potash-lye may also be added with advantage. After a time (usually between 3 and 6 hours), when the grinding and amalgamation of its contents are considered complete, one of the hand-hole covers is taken off, and the contents of the barrel washed out into the sluice-box, the bulk of the mercury being allowed to remain in the barrel; the sluice-box should contain a length of copper-plate and a set of riffles. If it is thought advisable to further treat the barrel tailings, they may be allowed to flow into the launders leading to the concentrators. When all the fine sands are thus washed out, the hand-hole is closed, the barrel turned with its man-hole upwards, and a fresh charge thrown in. The cover of the man-hole is then again fastened on, and the process repeated. When the mercury in the barrel is considered to be sufficiently charged with gold, the barrel is turned with its man-hole pointing downwards, a mercury pail is placed beneath it, and the contents of the barrel collected in the latter. The power required to drive a barrel continuously is about $2\frac{1}{2}$ I.H.P.

Batea.—In large mills much of the manual labour of the clean-up may be saved by the substitution of the mechanical batea for the usual hand prospecting pan. This batea, as manufactured by the Risdon Ironworks, San Francisco, is shown in Figs. 111 to 113. It consists of an oval or round shallow pan of cast-iron about 3 feet 6 inches or 4 feet in diameter, and about 3 to 4 inches

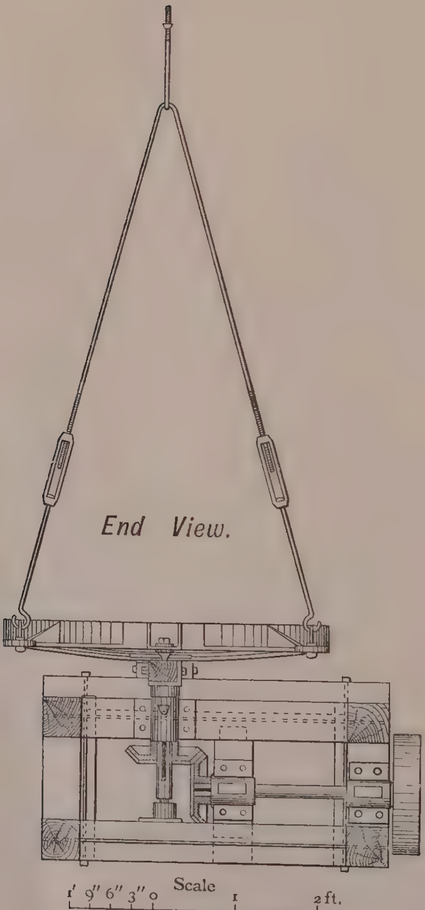
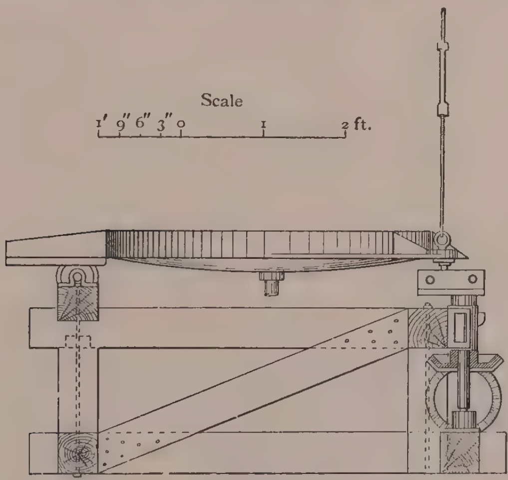


FIG. 111.

deep, with a rounded bottom and a plug in the centre. This pan is supported at one end on a roller, and

suspended at the other by a couple of rods of light round iron, this mode of suspension allowing it great freedom of motion. At the suspended end there is a small vertical crank driving a pin, which forms part of the pan itself, at a rapid speed, so that the pan receives a gyratory motion, not unlike that used in panning-up amalgam in the hand-pan, and which serves admirably to collect the



Side View.

FIG. 112.

quicksilver and amalgam in the bottom of the shallow cast-iron pan, the lighter materials finding their way to the sides. The power required to drive the batea is about 1 I.H.P.

Treatment of Amalgam.—The amalgam obtained off the plates throughout the month's run is usually stored in a special safe until the completion of the clean-up. The total mass of amalgam collected may accordingly

consist of hard amalgam off the inside and outside plates, softer amalgam from the mercury wells, amalgam from the clean-up, which may be hard, but is more often got in a pasty state, and pasty or semi-fluid amalgam from the barrels or other sources. Attached to the mill there should be a small mercury-room in which all operations connected with the treatment of mercury or amalgam

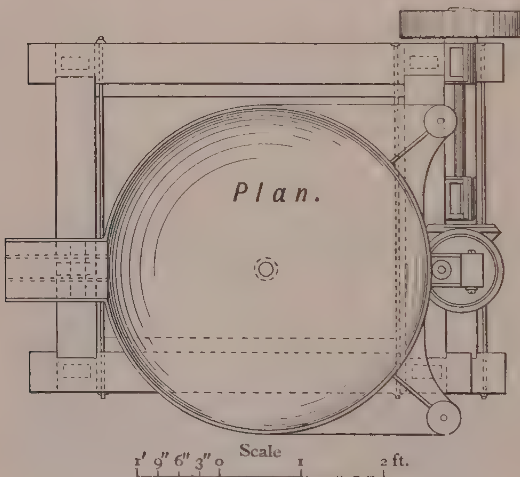


FIG. 113.

should be carried out. The amalgam safe is best placed in this room, securely fastened to the floor or to a main portion of the structure, unless it is a very heavy one. The room should have a sink and water supply, and a strong work-table. The top of this table should consist of a single slab of stone, or of a stout hard-wood plank 3 inches thick. It should have a rim a couple of inches deep all round its edges, and just inside this there should

be a groove leading to a small depression in one corner; sometimes the whole table has a slight inclination towards this depression. Cups and pails of enamelled ron, amalgam knives, a plentiful supply of buckskin and canvas for squeezing amalgam, should all be at hand, together with the requisite chemicals; a good strong balance capable of carrying 2,000 ounces and turning with $\frac{1}{100}$ ounce, together with the necessary set of weights (all preferably made of some substance other than brass) complete the equipment of this room. The amalgam from the plates should generally be clean enough to need no further treatment; it may be sprinkled with a little mercury if very hard, so as to make it slightly pasty, well kneaded, and made up into balls between 20 and 50 ounces in weight. If, on the other hand, it is too soft, that is to say, if it contains too much free mercury, it has to be squeezed.

Squeezing Amalgam.—This is done by pouring the necessary amount of soft amalgam into a strong, sound piece of buckskin or chamois leather, which has previously been well soaked, or into a bag made of fine canvas. The canvas used for this purpose should be strong but not too coarse; the quality which is made for the smaller sails of yachts is best suited to this purpose. It should be thoroughly soaked before use. The free ends of the leather or canvas are then grasped in the left hand just above the point occupied by the amalgam and twisted round so as to prevent the latter from escaping; the whole is then immersed in water contained in a mercury pail or pan, and the globular lump so formed twisted strongly with the right hand; by this means considerable pressure is put upon the amalgam contained in the skin or canvas, and the fluid free mercury is squeezed through

the pores of the latter, until only a ball of hard amalgam remains behind, all the superfluous mercury having been squeezed out of it. Several forms of mechanical mercury-squeezers have been invented, but none have yet proved sufficiently successful to supplant hand-squeezing. One of the best known is the hydraulic separator, which consists of a pail-shaped cast-iron vessel, the bottom of which is formed of a piece of buckskin or canvas, securely tied or otherwise fastened to the vessel. The amalgam which has to be squeezed is then poured in, and a cover is screwed on, into which is fastened a small water-pipe connected either with a small force-pump, or with a tank at some considerable elevation. By either of these means strong, hydraulic pressure is exercised on the surface of the amalgam which tends to drive the free mercury through the pores of the leather. Amalgam, however, cannot be squeezed dry by this apparatus so well as by hand, and it is accordingly very little used. A more modern device consists of a hydraulic ram, working into a cylindrical cage in which the amalgam is placed contained in a strong canvas bag.

It has already been pointed out (page 85) that when pasty amalgam is squeezed, hard amalgam of a more or less definite composition is left behind, whilst the escaping mercury is not pure, but consists really of a saturated solution of gold amalgam in mercury.

As already mentioned (see page 86), I have found as the result of numerous experiments on this subject that the amount of amalgam thus dissolved increases with an increase in temperature of the amalgam treated.

The following table gives the results of two experiments on this point; in each case the squeezing was as complete as possible by hand, but it must be remembered

that it is very difficult to ensure that the conditions as regards pressure shall be exactly uniform.

Temperature of Squeezing.	Gold dissolved in the squeezed Mercury in parts per 1,000.	Equivalent to Amalgam dissolved in parts per 1,000.
17° C.	0·57	1·72
51° C.	1·73	3·81
18° C.	0·46	1·93
72° C.	1·10	3 88

At the same time the residual amalgam was found to be proportionally richer in gold, the higher the temperature of squeezing :—

Amalgam	squeezed	at 72° C.	retained	282·43	parts of gold	per 1,000
,,	,,	,, 50° C.	,,	274·97	,,	,,
,,	,,	,, 18° C.	,,	238·76	,,	,,

As it is preferable that the escaping mercury should retain a minimum of gold, it is accordingly advisable to keep the water in which the squeezing is done as cool as possible.

Further experiments on the differential squeezing of small parcels of amalgam gave the curious result that the richness of the escaping mercury is not uniform from beginning to end of the squeezing, but that the first and last portions of the mercury are the richest in gold. The results of two such differential squeezings are quoted on p. 444.

It will be noted that these two experiments were performed on amalgams of distinctly different character, the first being derived from very finely divided gold, which yielded amalgam containing only 22·15 per cent. of gold, whilst the second lot of amalgam contained 35·955 per cent. The same remark as to the difficulty of securing

	Weight of Mercury taken.	Weight of Gold got.	Gold in parts per 1,000.	Equivalent to Amalgam in parts per 1,000.
1	685·6 grs.	0·518 gr.	0·755	3·41
2	693·5 ,,	0·441 ,,	0·636	2·87
3	1,139·0 ,,	0·686 ,,	0·602	2·72
4	1,077·0 ,,	0·980 ,,	0·909	4·10
5	637·0 ,,	0·881 ,,	1·226	5·53

No. 2.

	Weight of Mercury taken.	Weight of Gold got.	Gold in Parts per 1,000.	Equivalent to Amalgam in parts per 1,000.
1	493·3 grs.	0·705 gr.	1·429	3·97
2	841·2 ,,	1·068 ,,	1·270	3·53
3	407·2 ,,	0·540 ,,	1·326	3·68
4	492·7 ,,	0·697 ,,	1·414	3·93
5	401·0 ,,	0·547 ,,	1·365	3·40
6	370·6 ,,	0·570 ,,	1·538	4·27

uniformity of mechanical conditions applies to these results, as to the first batch of experiments, but I have corroborated them repeatedly and have no doubt as to their general accuracy, although I must admit that I cannot give any satisfactory explanation of the observed facts. It is easy enough to understand, and would *a priori* be expected, that the last portions of mercury squeezed out should be the richest in gold, but it is difficult to understand why the first should be richer than those immediately succeeding it. The practical lesson to be drawn from these experiments is that it is not advisable to carry the squeezing to the furthest possible point, but rather to leave the last traces of free mercury in the amalgam to be expelled by subsequent treatment in the retort.

Cleaning Amalgam.—The amalgam obtained from other sources has to be cleaned; indeed in many mills all the amalgam, from whatever source obtained, is thus treated. This operation is performed in the clean-up pan. This machine almost exactly resembles a small amalgamating pan, Fig. 102; its diameter is usually 3 feet but sometimes as much as 4 feet. The muller is furnished with wooden shoes instead of iron or steel ones, and the continuous muller is sometimes replaced by four arms carrying blocks of wood at their lower extremities, somewhat like a small settler. The usual speed of the muller is about thirty revolutions per minute, the power required to drive it at this speed being about 2 I.H.P. The pan is half-filled with water; from 200 to 1,000 ounces of amalgam are charged in, and from 20 to 150 lbs. of clean mercury. If the presence of grease is suspected, a little strong potash lye may be added to the contents of the pan, but cyanide of potassium, which is sometimes used, should be avoided. The lumps of amalgam are soon broken up, and form a uniform pasty mass, any sand or other impurities present rising to the surface. When the operation is complete, the muddy water, with as much sand as possible, is run off through a series of plug-holes. The muller is then stopped, and the contents of the pan drawn off into mercury pails; pieces of iron floating on the surface are removed by the magnet, and sand, &c., is washed off by means of a small stream of water. The surface of the mercury is usually found to be very foul. This is due to the presence of base metals which have found their way into the amalgam. The surface of the mercury is skimmed by means of a piece of wet flannel, canvas, or india-rubber belting, until it remains quite bright, the skimmings being put aside in a special pail

for separate treatment. The clean pasty amalgam is then squeezed as already detailed, and made up into balls ranging from 30 to 60 ounces in weight, ready for retorting.

Retorting.—The principle of this operation consists in heating the amalgam to a temperature above the volatilisation point of mercury, when this metal distils over, leaving the gold in a loose cellular state, in which it is known as gold sponge, the mercury being condensed by appropriate cooling apparatus. Practically there are only two forms of retort in use in gold mills, the pot retort for small quantities, say up to 1,000 ounces of amalgam, and the cylindrical retort for larger amounts. Pot retorts are made in various sizes to hold from 250 to 1,000 ounces; the former are about 6 inches deep inside, and the latter 9 inches, the diameter being about $\frac{2}{3}$ of the depth. The usual shape is shown in Fig. 114. It will be seen that the retort consists essentially of two parts, namely the body and the cover, the delivery-pipe being screwed into the latter. These retorts are made of cast-iron and carefully turned inside. It is advisable to have the inside of the cover turned as well as the body; the joints between these two should be very accurately turned and be as true as possible. It is advisable to have a V-shaped projection on the face of the flange of the body which fits into a corresponding annular groove in the flange of the cover. A piece of good wrought iron piping is screwed into the

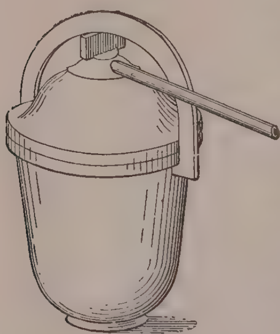


FIG. 114.

cover, its other end screwing into a stout Liebig condenser. This condenser consists of a pipe (of the same diameter as the retort outlet pipe) which passes axially through a short piece, 2 to 3 feet long, of a wider pipe so that an annular space, closed at both ends, is left all round the central pipe. An old mercury bottle answers capitally for the outer pipe. Two smaller pipes communicate with the top and bottom respectively of the annular space, water being supplied through the lower and escaping through the upper one. In its passage up it completely cools the heated vapours that are passing down through the small central pipe. The retort is usually supported on a strong iron tripod, and may have a special furnace for heating it, although this is not necessary. Often a fire is built on the ground under and round the retort, the heat being concentrated upon the retort by laying a few bricks, or supporting some pieces of sheet-iron round it. The assay furnace (page 516) does admirably for heating the retort; sometimes a smith's hearth is used, but this practice is not to be recommended. There are several methods by which the cover is secured to the retort body. The flanges may be clamped together, or they may be bolted together by three bolts passing through them, cotter bolts are better than screwed ones. Sometimes a semi-circular bale is used catching under the flange of the body, whilst a strong set-screw or a wedge presses on the top of the cover. Of all these plans the last, illustrated in the above figure, is the best, as it is the least apt to be injured by the action of the fire. Before charging the amalgam into the retort, its inside should be coated with some substance to prevent the gold sponge from sticking to it. It may be well rubbed in with chalk or whitening, but the best coating consists of equal parts of

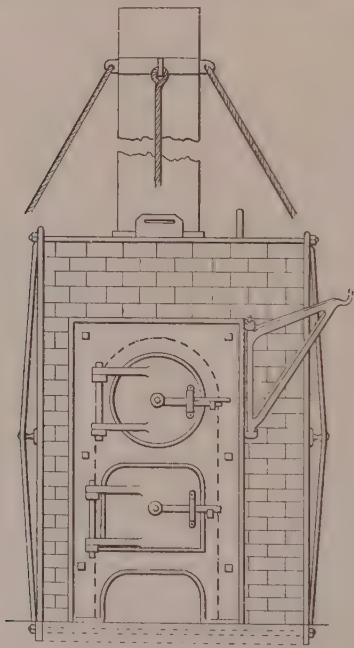
finely ground fire-clay and graphite made up into a thin paste. The retort should be washed out with this, and then put in a warm place so as to dry the coating. The same mixture worked up with water to the consistency of cream may be used for a luting between the flanges of the body and cover. It is also advisable to coat the outside of the entire retort with a similar mixture, to which a little fine asbestos has been added, as this preserves the retort from burning out rapidly. The balls of amalgam should be broken into two or three pieces each, and piled loosely upon each other in the retort, which should never be more than two-thirds full. A disc of stout asbestos millboard, just about $\frac{1}{8}$ inch smaller in diameter than the top of the retort, should then be dropped in, a thin layer of lute spread on the flange, the cover put on, turned backwards and forwards a few times to ensure a tight fit, and then secured in its place. The retort is next placed on its tripod ready for heating and the condenser attached.

The end of the condenser pipe should be a few inches above the surface of water contained in a mercury pail, and should on no account dip below it. A strip of canvas should then be tied round the end of the discharge pipe so as to form a kind of loose tube dipping into the water, but care must be taken that this tube is not air tight. Sometimes a canvas or india-rubber bag is attached to the end of the pipe, but the above arrangement is preferable. A fire of small billets of wood, bark, or small brushwood is then built under and about the retort, the fire being preferably so arranged as to burn from above downwards, and the temperature very gradually raised until mercury begins to distil over. The heat of the fire must then be moderated so as just to keep the mercury distilling over in a gentle stream, but no more. The

temperature of the retort should never approach redness as long as any mercury at all distils over. When no more comes over, the heat must be raised to redness, and kept at this point for a few minutes. The fire can then be removed, and the retort allowed to cool. The entire operation takes two to four hours as a general rule. It must not be forgotten that as the mercury distils over and collects in the pail, the level of the water in it will rise, so that a little must be dipped out from time to time to prevent its rising above the mouth of the pipe. When the retort is cool enough to handle, the cover is taken off, the looting scraped off the flange, and the disc of asbestos board lifted out. The retort is then inverted over a piece of stout paper or a prospecting pan, when the sponge will drop out in one coherent retort piece, if the operation has been properly carried out. If the retort has been badly coated, or the heat too great, the sponge may adhere in places to the retort. It may then mostly be dislodged by a few taps of a hammer on the bottom of the retort, or, in extreme cases, a light hammer and chisel may have to be used. The object of the disc of asbestos board is to prevent spirting, and the mechanical carrying off of any of the amalgam in the vapours of mercury. The sponge, when perfectly cold, is weighed and cut up with a hammer and chisel preparatory to melting.

The cylindrical retort is shown in Figs. 115 and 116. This is usually built into a special furnace as shown. It consists of a cylinder of cast-iron 9 to 12 inches in diameter and 3 to 4 feet long. Into one end is screwed a delivery-pipe which communicates with a condenser as in the case of the pot retort. In this case it is, however, on a larger and more substantial scale, and is usually a fixture. The other end of the retort is closed by a door

which is secured by bolts or wedges like the cover of the pot retort. Underneath the retort is a fire-place usually about 1 foot × 2 feet when wood is the fuel used, and smaller when coal is employed. This is fitted

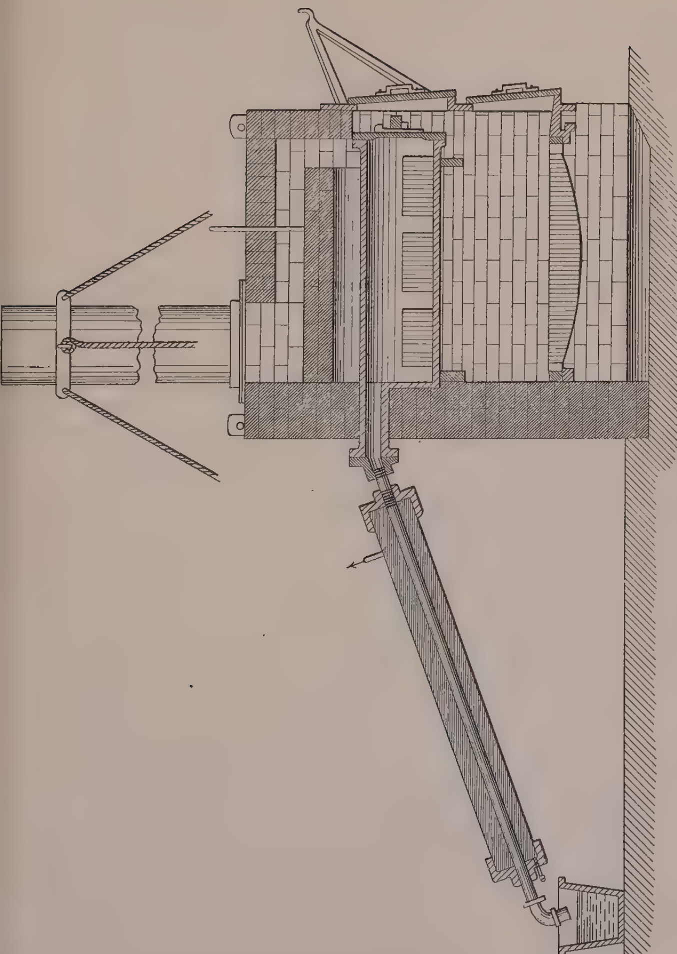


Front Elevation

Scale, ⅜″ = 1 ft.

Fig. 115.

with a cast-iron door, as may also be the ash-pit below it, whilst the stack is furnished with a suitable damper. Sometimes special provisions are introduced to provide for a uniform distribution of the heat to all parts of the retort. There are usually three or four trays of stout



LONGITUDINAL SECTION

Scale, $\frac{3}{8}'' = 1$ ft.

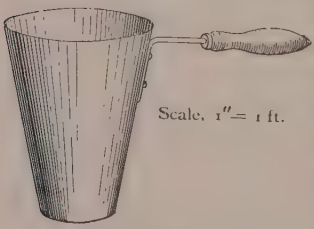
FIG. 116.

sheet iron which fit into the retort and carry the amalgam; these should be of such a size as to leave empty about the fifth of its length furthest from the door. These trays are coated in the same way as already described for the interior of the pot retort, they are charged in a similar way, and pushed into their places, being lifted up by the help of a small crane, which is mostly attached to some portion of the iron framing of the furnace. The cover is then by the same means lifted into its place, luted, and secured. Firing up is then commenced with the same precautions as recommended in the case of the pot retort, namely, very gradual heating a first, until the mercury commences to distil. At this point the temperature is kept as nearly as possible stationary, until all the mercury has distilled over, when it is raised to redness. This usually takes about six hours. When the retort is cool the door is lifted off by means of the crane, the trays drawn out, and the sponge turned out from them. Various minor variations have been made in the form of the retort; sometimes the outlet pipe is at the top, sometimes it is central. The former arrangement has the advantage that it is less liable to get stopped up during the operation; the latter that when the bottom part of the retort commences to bulge and become deformed, owing to the greater heat to which it is exposed, the retort can be turned upside down, and its life thus prolonged. Retorts may be oval or circular in section; some are cast with several projecting ribs round them so as to strengthen them, and the better to keep them in shape. Most mills, however, use the plain cylindrical retort. With proper care these retorts last for some years, usually two to three.

When retorting is carried out with proper precautions, the loss of mercury is exceedingly small; at the same time, too, the operator runs very little risk of salivation. It is obvious that these two dangers are intimately connected, as both mean that mercury vapour is being allowed to escape into the air. The most frequent cause of this is taking the cover off the retort before it has thoroughly cooled down; it is very rare that the retort cracks in use. Retorted mercury always retains, as already said, a trace of gold equal to about 0·005 part per thousand, or, say, one grain to about thirty pounds of mercury. This mercury should be purified in the usual way, as it may carry traces of lead, zinc, &c., and is then fit for use again.

Melting.—This is always performed in crucibles heated in a pot furnace. Sometimes, in very large mills, a special furnace is built for this work, in which case it is usually built against the side of the retorting furnace, and opens into the same stack. The assay furnaces described on page 516 do thoroughly well for melting furnaces, provided that it is not considered necessary to melt very large quantities at a time, and in this case the square wind furnace, shown in Fig. 122, is still the best type of melting furnace, its dimensions being increased so as to take the requisite crucible. The top of the furnace should not be too high above the ground, should be kept level, and covered with a flat cast-iron plate. It is an advantage to have this plate so large that the ingot moulds can be set upon it. Plumbago crucibles are usually employed for melting gold when no fluxes are required; when fluxes are added, either good clay crucibles, such as the Battersea pots, or even better, the Salamander crucibles, made by the same firm (the Morgan Crucible Company,

Limited), should be used. These crucibles run from about 3 to 12 inches in height by $2\frac{1}{2}$ to $8\frac{1}{2}$ inches in diameter, outside measurements. As a general guide, it may be taken that a crucible 10 inches in height will hold about 1,500 ounces, whilst one $6\frac{1}{2}$ inches high will hold about 300 ounces. The furnace to heat any given crucible should have a diameter about one and a half times as great as that of the crucible, and be from 3 to 5 inches deeper. By far the best fuel to use is coke, and it is worth while paying a liberal price for it to ensure its being of good quality. Charcoal can be used in case of need; hard-wood charcoal is the best, and it should be screened, all that passes through a $\frac{3}{4}$-inch riddle being rejected. When charcoal is used a good draught is indispensable, and, if required, a blast may even be used.



Scale. 1″ = 1 ft.

FIG. 117.

The crucible should always be warmed before use. Salamander crucibles do not require annealing, but for ordinary plumbago crucibles this is indispensable, and even for clay pots it is advisible. In order to anneal a crucible it should first be thoroughly dried in the ash-pit of the melting furnace, of a boiler fire, or in some similar convenient place. It is then placed mouth downwards on a cold fire, and the heat gradually raised until the crucible is red hot. The crucible is then taken out, any bits of fuel or other dirt that may have got in shaken out, and the crucible arranged in the fire, with its mouth upwards ready to receive the sponge. The crucible should be placed upon a stand made of fire-clay about 3

inches deep and a little larger in diameter than the bottom of the crucible; a piece of firebrick chipped into shape makes a very good stand. When a small-sized crucible is being used in a coke fire this stand may be dispensed with, as this fuel burns away but slowly and makes a firm bed for the pot. But in the case of a quick-burning fuel like charcoal it is absolutely necessary to use a stand to support the crucible. Needless to say, the stand must be put in its proper place, resting firmly on the fire-bars, before the fire is lit. When the crucible has been put into place it is covered with its cover, and the fire is made up, fuel being packed all round it. The fire is allowed to burn well through, and the sponge is then charged in by means of the sheet-iron funnel shown in Fig. 117, or, in its absence, a large-sized assayer's scoop may be used. The sponge will then melt down pretty fast; when completely melted, a fresh lot is charged in, and so on, until the full charge has been transferred to the crucible. Care should be taken in charging never to fill the crucible more than three-fourths full with the sponge, and the molten gold will then never rise quite as high; of course the sponge occupies a very much larger bulk than does the molten gold. If the fire at any time burns hollow, it must be well poked down and more fuel added, the crucible being kept closely covered at such times. When charcoal is used, this has generally to be done several times during the course of a melt. Small mills have at times no proper melting furnace; in this case the crucible may be heated upon the smith's hearth. The crucible rests on a brick so placed over the *tuyère* that the blast shall rise up on either side of it. A rough wall is then built round it, made of clay, bricks, pieces of old iron plate, &c. The interior is then filled up with fuel,

and the fire urged by means of the bellows till the gold is thoroughly melted. This rough-and-ready method requires, however, considerable care lest the crucible should crack and the gold run out; there is, however, little fear of ultimate loss, as the gold can always be panned up again from the ashes of the hearth.

When the sponge is clean and free from base metals, it is best melted by itself without the addition of any flux. This may be judged of by the colour of the sponge; when it is of a bright yellow colour it needs no flux, but if it is greyish or blackish it needs refining, otherwise the resulting ingot may be brittle. In the first case the sponge is charged directly, as already explained, into a blacklead or Salamander crucible. When the melt is complete, the surface should be clear, brilliant, and of a greenish yellow colour. There may be a little scum floating on the surface, and this can be scraped off by means of an iron rod the end of which is flattened out. The crucible is then drawn out from the fire; for large pots carrying 500 ounces and over it is advisible to use "basket tongs" (Figs. 118 and 119). An iron link should be slipped over the handles to keep the tongs from slipping when grasping the crucible. As soon as it has been pulled out, the crucible is released from the basket tongs, any adhering cinders are knocked off, and the molten gold is poured in a thin steady stream into the ingot moulds, the crucible being grasped by the assayer's tongs shown in Fig. 126, page 536, or by those shown in Fig. 120.

When the sponge is impure and needs refining, it has to be melted down with a refining flux. The most suitable flux consists of a mixture of about equal parts of nitre and carbonate of soda with a little borax.

Sufficient of this should be charged with the first lot of sponge to make a layer about half an inch deep on the surface of the molten gold; or else only about half this amount may be added at first and a little more with each fresh lot of sponge. Sometimes brittle gold is toughened

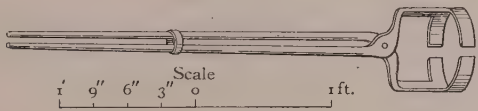


FIG. 118.

by throwing small pieces of sal-ammoniac on to the surface of the molten gold. When the gold is ready for pouring, the slag should be pushed back from the spout of the crucible, so that the first portions of gold may run free from slag, which will form a layer on top of the ingot. The moment the gold has set and whilst the slag is still liquid, the ingot should be tipped from the mould into a bucket of water, when the slag will come away readily from the ingot; in case any should adhere it can be

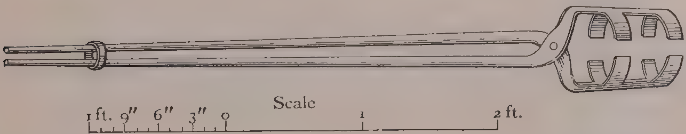


FIG. 119.

loosened by means of a little warm dilute nitric acid; hydrochloric acid must not be used, as gold would be dissolved by it on account of the nitre in the slag.

Ingot Moulds.—These are best made of cast-iron of the shape shown in Fig. 121. The inside should be

carefully planed and finished, and all the angles neatly rounded. The name or initials or any other distinguishing mark of the company owning the bullion may be cast in the bottom of the mould; care however must be taken that these letters do not form any sharp angles, which would be likely to keep the ingot from dropping out readily. It is not usual to cast ingots larger than 1,000 ounces. A set of moulds for 50, 100, 250, 500, and 1,000 ounces is sufficient for the requirements of most mills. An ingot mould 3 inches deep, 12 inches by 4 inches at the top and 11 inches by 3 inches at the bottom, will just about hold an ingot weighing 1,000 ounces.

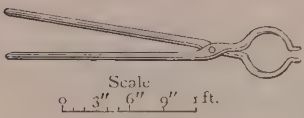


FIG. 120.

When casting ingots of gold melted without flux, it is best to cast them under oil. Enough vegetable oil (such as olive oil) to form a layer $\frac{1}{4}$ inch deep is poured into the mould, which must be heated up to the boiling-point of the oil; it is then ready to receive the molten gold. When there are fluxes with the gold, oil should not be used. The ingot mould must then be very thoroughly blackleaded and heated so as to be perfectly dry before gold is poured into it.

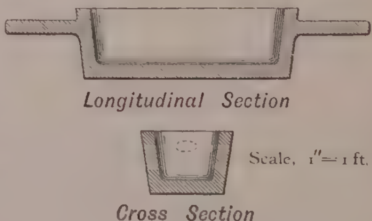


FIG. 121

Breakage of Crucibles.—It occasionally happens that a crucible cracks during the progress of a melt, although this is a rare accident if the precautions above enumerated are observed. As soon as a crucible is seen

to be cracked, it is drawn from the fire and allowed to cool, or, if possible, the whole or part of its contents are poured, best into water so as to granulate them. The fire is then allowed to burn itself completely out. The inside of the furnace and the fire-bars are scraped down and the contents of the furnace and ash-pit carefully brushed out. They, together with the cracked crucible, are pounded down and passed through a sieve of 40 holes to the linear inch, any lumps of gold met with being picked out. The pounded material is then panned up, when the whole of the gold should be recovered. It must be melted down with nitre and borax, to free it from iron, with which it is pretty sure to have alloyed to some extent.

Losses in Melting.—These should be very small indeed and are due principally to spirting, or the projection of minute prills of gold out of the crucible, or their adherence to the crucible. They may be reduced to practically nothing by sending all the residues obtained in the melting-house, such as old crucibles, furnace cinders, flue dust, slags, &c., back either to the stamp-mill or the amalgamating barrel, and this is a precaution which should never be omitted. Similarly, old leathers or pieces of canvas used in squeezing amalgam should be put aside, burnt from time to time, and their ashes also subjected to amalgamation.

Treatment of Skimmings.—The skimmings from the surface of the mercury and amalgam (page 445) always contain a notable proportion of gold. They should be transferred to a small pot retort, which they must not more than one-third fill, and retorted slowly. The temperature must be kept down, and should not be allowed to reach redness, even at the end of the operation, as the

residue is sometimes very fusible, and might stick to the retort if strongly heated. If the amount of skimmings is very small, these may be transferred to a crucible, which is then put into the retort, being packed all round and kept in its place by means of coke dust or sand, and the retorting thus performed, when there will be no danger of injuring the retort even if the temperature rises above redness. A black scoriaceous mass is usually obtained, which may be fused repeatedly with nitre and borax, when a button of clean gold will ultimately be produced.

Fusion of Precipitated Gold.—Precipitated gold, as obtained from chlorination or cyanidation, has at times to be melted. It should be moistened with a strong solution of borax, forcibly compressed in some appropriate machine whenever practicable, thoroughly dried, and then charged into the crucible just like sponge. Some workers prefer to omit the borax, and if the gold powder can be compressed by a powerful press this is the better plan. The only danger is that some of the finely divided gold may be carried off by the furnace draught, and if the sponge is not thoroughly dry, loss from this source is sure to occur. The proper treatment of gold slimes, as obtained by the precipitation of cyanidation liquors by means of zinc, has already been considered (page 417).

CHAPTER XIV

MODES OF TREATMENT—COST OF MILLING—GENERAL CONSIDERATIONS—LABOUR—MILL SITE AND MILL BUILDING—POWER—ILLUMINATION—RETURNS

Modes of Treatment.—It is now fairly evident what methods of gold extraction may be looked upon as those that should be recommended in any special case. Obviously, different ores require different modes of treatment, though there is no sound reason that can justify the very wide divergencies of milling practice in various parts of the world. Moreover, it is not sufficiently recognised that the proper method to be employed can always be ascertained beforehand by laboratory tests which can determine the amount of gold that can be extracted by the various operations already considered. If the bulk of the gold present is not "free," the ore should not be treated by milling, direct cyanidation, direct chlorination, or smelting being resorted to, so as to suit the nature and richness of the ore and the circumstances of the locality. It is only when a considerable proportion of the gold can be extracted by amalgamation that milling should be employed at all, and it is only to such ores (constituting the vast majority of gold ore) that reference

will be made here. There are two broad classes into which all milling ores may be divided, low-grade and high-grade ores, the limit between them being taken at present as about 7 dwt. of gold to the ton. Sometimes it happens exceptionally that a low-grade ore is treated by high-grade methods, or a high-grade ore by low-grade methods, but such instances are rare.

Low-Grade Ore.—The treatment of low-grade ore is one that may be considered as having originated entirely in the Western States of North America, where the ingenuity of gold miners has been aided very powerfully by great natural advantages, such as cheap timber and fuel, and, above all, a plentiful supply of water at high levels. The essential condition of success in the treatment of low-grade ores is that the ore shall be kept in continuous movement by purely automatic methods, so that from the time the ore leaves the mine cars until it flows out of the mill in the form of waste tailings, it shall never be subjected to handling, whilst all the operations that it undergoes shall involve a minimum of manual labour. In this connection, too, it is important to remember that the object of the mill man is not merely to extract as much gold as possible from the quartz, nor even always to extract it as cheaply as possible, but to make the largest possible cash profits. Thus it very often pays to put a larger quantity of ore through the mill, even though a little gold be thereby left in the tailings that might be extracted by a slower rate of working, or, although the cost of treatment per ton be slightly increased, provided that the amount of ore to be mined is unlimited for practical purposes, and can be won very cheaply. Thus, for example, a company having very large reserves of ore would prefer to put

through 200 tons of ore per day at a profit of 1*s*. 9*d*. per ton rather than 100 tons of ore at a profit of 2*s*., although, as a matter of fact, an increase in the quantity of ore crushed nearly always means a reduction in the cost of crushing per ton, as will presently be seen.

One of the best types of mill for treating low-grade ore is to be found in the Black Hill region of Dakota. The ore there worked yields about 4 dwt. of free gold per ton; this is milled by means of 850 lb. stamps, making on an average about 80 10-inch drops per minute; the depth of discharge is about 10 inches, and the width of the screen apertures 0·024 inch. The crushing capacity of these mills is 4½ tons per head per 24 hours. Inside amalgamation is practiced, about 60 per cent. of the total yield being thus obtained; a short length of blanket sluice is placed below the copper-plates and the pulp runs through amalgam traps. No attempt is made to further treat the tailings, whose average gold contents are about 16 grains per ton, mostly locked up in sulphurets, which latter amount to 3 per cent. The district affords a first-rate example of large mills treating low-grade ore for free gold only and treating it at a profit.

Another system of ore treatment, which is perhaps the most widely employed, and which may be looked upon as the typical modern Californian method, is that of inside and outside amalgamation, followed by concentration, and subsequent chlorination of the concentrates. This is the system adopted in that highly successful mine the Alaska-Treadwell, in Alaska. The ore from this mine yielded in 1897–1898 at the rate of 2 dwt. 7 grs. of gold per ton, 1 dwt. 12 grs. being free, and the remainder being derived from the concentrates by chlorination; the ore yielded 1·6 per cent. of concentrates of an average

assay value of 2 ozs. 3 dwt. There were 240 stamps of 900 lbs. each, the width of the screen apertures being 0·022 inch, and the capacity of the mill 3·05 tons per head per 24 hours. The neighbouring mill of the Alaska Mexican Gold Mining Company, which is worked on the same principles and under the same management, is doing almost bettter work. This mill contains 120 stamps weighing 1,020 lbs. and making 96 8-inch drops per minute; its crushing capacity is 3·79 tons per head per 24 hours. The ore yielded in 1897 nearly 2 dwt. 3 grs. per ton, the tailings carrying 4·5 grains of gold to the ton; of the above yield 1 dwt. 10 grs. was free gold, the remainder being obtained by chlorinating the concentrates; the production of these latter was 1·8 per cent. of the ore, and their assay value 1 oz. 10 dwt. 12 grs. per ton. In spite of the low grade of the ore, the profit for the above year was no less than 25 per cent of the production of gold. The new mill of the Alaska-Treadwell Gold Mining Company consists of 300 heads of 1050 lb. stamps, making 101 7-inch drops per minute and crushing at the rate of 4·45 tons per day. The concentrates are now no longer chlorinated but shipped to the Tacoma smelter. The ore yielded free gold to the value of $1·2213 (about 5*s*. 1*d*.), and concentrates worth $0·7362 (about 3*s*. 1*d*.), making altogether $1·9575 (about 8*s*. 2*d*.) per ton of ore. This company now owns five mills containing 880 heads of stamps.

Most of the large mills in California work upon this system, which seems to be the most suitable hitherto devised for low-grade ore carrying sulphurets. The concentrator now most generally used is the belt vanner in some form, the most popular being probably the Frue, but tables of the Wilfley type are also coming largely into

use. A few mills are, however, beginning to appreciate the advantages of sizing before concentration, and it is quite likely that this method will be more largely adopted in the near future. In many plants, large canvas tables now follow the concentrating appliances in order to save the very finely divided sulphurets. Occasionally, as for instance at the Drumlummon mill, Montana, inside amalgamation is not used. This mill has 60 stamps of 750 lbs. each, making an average of 95 8-inch drops per minute. It is said that the use of mercury in the battery is here objectionable, so that the pulp is discharged on to amalgamated tables, followed by Frue vanners, and these again by launders furnished with riffles.

High-Grade Ore.—The best example of the treatment of these on modern scientific principles is to be found in the Witwatersrand district of the Transvaal. Here the stamps are usually 900 to 1,000 lb. stamps, making from 90 to 100 4- to 7-inch drops per minute, and crushing through screens having apertures of an average width of 0·024 to 0·035 inch. The average crushing capacity of these mills is about $4\frac{1}{2}$ tons per day, some, however, doing as much as $5\frac{1}{2}$ tons. During 1897, the average number of stamps running was 3,567, which worked for 329·69 days and crushed 5,325,355 tons, equal to an efficiency of 4·53 tons per head per 24 hours. The process adopted is inside and outside amalgamation and at times concentration, generally on belt vanners, followed by chlorination or more rarely cyanidation of the concentrates; the tailings are more often not concentrated, but treated together with the sulphurets. They are run into settling pits or large Spitzkasten, which separate the sands from the slimes, each being

treated by special cyanidation processes. The results obtainable by this method in its highest stages of development are well shown in the excellent reports issued annually by some of the Witwatersrand mines; those of the Crown Reef Gold mining Company, Limited, for 1897–98 may be quoted as typical. The mill consisted of 120 head of 995 lb. stamps, which ran for 334·9 days, crushing 185,179 tons at the rate of 4·608 tons per head per 24 hours. The following table summarises the results obtained:—

	Percentage treated of ore milled.	Fine gold recovered per ton milled. Dwt. Grs.	Percentage of total amount recovered.	Percentage of total amount lost in residue from each operation.
Ore milled	—	8 6·715	57·806	—
Sands cyanided	71·799	3 1·127	21·272	6·696
Concentrates cyanided..	3·303	0 23·420	6·812	0·970
Slimes cyanided	22·116	0 12·452	3·622	1·561
Slags and bye-products.	—	0 1·379	0·401	—
Slimes not treated	2·782	—	—	0·855
Totals	100·000	12 21·095	89·913	10·082

The ore contained by assay 14 dwt. 7·759 grs. of fine gold to the ton.

A method which has yielded good results in local use is the so-called "Grass Valley" method, from its having been introduced in that district of California. Battery amalgamation was not employed, the pulp being allowed to run over blanket strakes, and the blanketings so collected being amalgamated by special amalgamating machines, such as the Atwood amalgamator, Eureka

rubber, pans, &c. Amalgamated copper-plates played a very subordinate part in this process, being only used to catch any free gold that may escape from the blankets. This process required a good deal of hand labour, and was not so perfect a gold-saving method as the more modern ones. Since about ten years this old method has been abandoned in favour of the typical Californian process, namely, milling in mortars fitted with inside copper-plates, followed by outside copper tables and belt concentrators.

An interesting variation on the above process is that of the old St. John del Rey Mine, in Brazil. This consisted in crushing, collecting the sulphurets upon blanket strakes, and amalgamating the blanketings in barrels, similar to those used for silver amalgamation in Germany. The stamps used weighed 640 lbs., making 75 12-inch drops per minute, the depth of discharge being 18 inches. They crushed two tons per head per 24 hours. A special pattern of reversible blanket strakes was employed. Closely allied to this method is the old practice at Clunes in Victoria. A very similar antiquated process is still in use at Berezovsk in the Urals. The mortars are of the old-fashioned low type; there are thirty 700 lbs. stamps making about 65 drops per minute; the screens have about 0·03 inch diagonal slots; the mill crushes about 55 tons per 24 hours. The pulp runs over blanket strakes, four to each 5-stamp battery, about 25 feet long and 1 foot wide, three being always in use, whilst the fourth is being cleaned up; these are followed by three similar strakes containing amalgamated copper-plates, in very bad order, which is hardly to be wondered at seeing that only two of these strakes are run at a time, the third being left dry. The blanketings are washed in a small flat buddle about 7 feet long, 4 feet wide, and

2 feet high. After the sands have been concentrated by washing, about two teaspoonfuls of mercury are poured in and worked into the sands with a wooden hoe: then the concentrates are washed off, leaving pasty amalgam behind. Of the total gold got 95 per cent. is from the blanketings and 5 per cent. off the plates.

Similar remarks may be applied to what is known as the Colorado system of gold milling as to the Grass Valley method. In its original form this process was applied to comparatively soft ores, carrying a large percentage (10 to 20) of comparatively low grade pyrites and very finely divided gold. The stamp is here used as an amalgamating machine even more than as a crushing one. The average weight of the stamps is about 600 lbs., working at the rate of 25 to 30 18-inch drops per minute. The screen aperture is usually about 0·02 inch, and the depth of discharge 15 inches. The result of this arrangement is that only about one ton is crushed per head in 24 hours, the cost of milling being proportionately augmented; the loss of mercury is also heavy. Inside and outside copper-tables are used, followed mostly by blanket strakes and end-shake concentrating tables. The process has developed itself slowly and gradually, but has changed far less within the last twenty years than in the case of Californian mill practice. It cannot be recommended for new countries, as it manifestly presents few advantages over the Witwatersrand system, and has many corresponding disadvantages. At the same time it must be admitted, that probably no other method (except perhaps the Hungarian system of stamp milling followed by treatment in Hungarian mills or Laszlo amalgamators) could extract so large a proportion of amalgamable gold. Whether milling is the proper

process to apply to such ores, or whether they should not rather be treated by the chemical or the smelting methods now in vogue in the Cripple Creek district, for example, is another question.

In Nova Scotia very good work is being done upon the lines of the Californian practice, which is closely followed.

In Australia, the most recent gold-producing colony, Western Australia, appears to be following Witwatersrand practice, though several of the mines are trying direct cyanidation; the character of the ores in depth has hardly been definitely ascertained, but it seems that the greater portion is not free-milling, and amalgamation is being largely displaced by other methods. In the older colonies there is but little to be learnt, the milling practice being in the great majority of cases far behind the Californian, though running upon the same general lines. Rock-breakers and self-feeders are conspicuously absent. The weight of the stamps is usually about 900 lbs. and the speed about 80 8-inch drops per minute; the screens are mostly punched iron and the average crushing capacity rather below 2 tons per head per 24 hours. Inside plates are very rarely used, but mercury is at times charged into the mortars. Outside plates are used to a considerable extent, but much reliance is still placed on mercury wells. Very varied forms of concentrators are used, end-shaking tables being in favour; these are often followed by blanket-strakes and the latter at times by canvas tables. The rough concentrates so obtained are still largely treated in Berdan pans, although in most large milling centres there are customs chlorination works, that buy and treat sulphurets at reasonable rates.

New Zealand practice is almost identical with

Australian. The high grade of the Australian ores and the coarseness and "freeness" of much of the gold are the probable causes of the comparative backward condition of scientific gold milling in Australia, another important factor being the comparatively small size of the mining claims in most of the Colonies, which but rarely admit of the establishment of large and well-equipped mills on the mines themselves.

Cost of Milling.—The cost of milling a ton of ore is influenced by so many different conditions that it is quite impossible to lay down any broad general rules. It is, of course, a matter of great importance to be able to estimate beforehand what the approximate cost of treating any given ore will be, and this task has often to be attempted. In preparing such estimates, the following points are those that need the most careful consideration.

I. *Nature of the Ore.*—It is evident that two ores of equal assay value may be of very unequal actual value, because the gold contents of the one may cost far more to extract than in the case of the other. Thus one ore might carry all its gold in the form of free-milling gold, and require no treatment beyond amalgamation, whilst another may need concentration, followed by elaborate after-treatment. One ore might carry its valuable ingredients in the form of coarse particles, whilst in the other they may be very finely divided, so that the second ore will need to be crushed far finer than the first, the natural result being that the same plant and staff will be able to crush less of the latter than of the former in the same time, the cost of treatment being correspondingly increased, whilst the loss of the slimes will be greater. A hard tough ore, again, will cost more to crush than a friable, easily broken one. Obviously the crushing

capacity of the mill will also greatly influence the cost of crushing the ore; thus a mill crushing at the rate of 2 tons per head per 24 hours will work at a tonnage cost very nearly double that of a similar mill crushing 4 tons per head. The cost of labour will be the same in either case, and the cost of fuel but little more, whilst the cost for mercury, spare-parts, &c., should vary almost directly with the tonnage crushed. There is hence a great economic advantage in running a mill at top speed; for example, a mill running at 60 drops per minute was found to crush 50 per cent. faster than the same mill run at 48 drops per minute, whilst the fuel consumption was very little greater. The fastest crushing compatible with good gold extraction is therfore invariably the best practice.

II. *Power Available.*—The cheapest form of power obtainable is water power, which costs very little more than the interest on the capital expended in constructing the necessary dams and water-courses and erecting the motors, the outlay requisite to keep these in repair being generally a comparatively small item. When water power is not available and steam has to be used, the quality and price of the fuel obtainable is a very important element. On an average a ton of quartz requires for its reduction 0·05 ton of coal or 0·07 cord of wood, these figures, ranging from 0·03 to 0·10 in the case of coal, and 0·03 to 0·12 in the case of wood, being influenced principally by the quality of the fuel. The importance of the cost and quality of fuel can therefore readily be appreciated.

III. *Labour.*—This is an item that may vary within very wide limits. Most of the labour required about a stamp mill is skilled labour, and this is often dear in

proportion as unskilled labour is cheap. For instance, in countries where low-priced native labour is employed for all purely mechanical purposes, trained white tradesmen generally command high wages. The practice of workmen, too, in different parts of the world varies greatly in respect of such matters as length of shift, and it is rarely possible to induce men to work except in accordance with the local customs, whatever these may be.

IV. *Size of Mill.*—This is, of course, governed by the available supply of ore, and is within certain limits one of the most important elements of the cost of milling. Thus it takes scarcely any more men to attend to a sixty-stamp than to a ten-stamp mill, so that the cost of labour per ton is but little more for the larger mill than one-sixth of that for the smaller mill. On the other hand, a 240-stamp mill will require nearly three times as many hands as a sixty-stamp mill; so that it may be considered that the limit of economy as regards mill labour is almost reached in a mill of sixty to eighty stamps. Of course, in other items of cost, such, for instance, as management, supervision, assay department, &c., the larger mill, whatever its size, costs scarcely any more than does the smaller one.

V. *Situation.*—As the cost of mill supplies forms a large element in the cost of milling, the comparative prices of these, which are largely determined by the cost of transport from the nearest sea-board to the mill, affect greatly the cost of the operation. The wear and tear of a mill amounts, as we have seen, roughly speaking, to some 2 lbs. of metal for each ton of ore crushed, and the freight on this quantity may readily amount to a very important item.

In addition to these principal points, there are usually

a number of others of smaller importance, depending upon local or accidental circumstances, which can scarcely be enumerated here, and which can only be discovered by investigating the actual costs of milling in neighbouring districts or in others where the conditions of work are as nearly as possible identical.

Actual Costs.—The actual costs of milling in a number of typical mills, taken from data obtained in practice and collected from various sources, are here tabulated, classified according to the respective countries; in the comparison of these due regard must be had to the points already enumerated.

North America.—In the table on page 474 are given the costs of milling in several mills in the United States, some of the smaller and worse-conducted establishments being included for the sake of comparison. It will be seen that the variations are within very wide limits. The slow crushing mills of Colorado (Gilpin County) naturally work much less cheaply than the modern mills of California; in the former the rate of crushing is now mostly between 1 ton and 1·25 tons per head per 24 hours. The cost of milling at the Hidden Treasure mill, Gilpin County, was \$2·42 (10*s*. 1d.) per ton, and in the Kansas mill in the same county \$1·47 (6*s*. 2d.) per ton in 1896; the latter item is made up of :—

	\$
Labour	0·62
Supplies, fuel, water, &c.	0·55
Tramming to mill	0·30

The North Star mill, Grass Valley, Cal., is an example of a mill doing excellent and cheap work in spite of the fine, and therefore relatively slow, crushing which its ore requires. It must however be noted that the figures given in the table over leaf include in this case the cost of con-

Name of Mine.	Locality.	Date of Statement.	Number of Stamps in Operation.	Tons crushed per Stamp per 24 hours.	Nature of Motive Power.	Costs per ton (of 2,000 lbs.) crushed. Labour.	Fuel.	Water (when paid for).	Mercury.	Shoes and Dies.	General Mill Supplies and Miscellaneous Items.	Total, (U.S. Currency).		Total, Sterling.	
						$	$	$	$	$	$	$	cts.	s.	d.
Alaska Treadwel.	Alaska .	1893	240	2·9	Steam.	0·2204	0·2712	—	0·0059	0·0504	0·1139	0	66·18	2	9·1
,, ,,	,,	,,	,,	,,	Water.	0·1880	—	—	0·0059	0·0504	0·1010	0	34·53	1	5·3
Alaska Mexican	,,	1895	60	3·68	$\frac{4}{7}$ time water. $\frac{3}{7}$ time steam.	0·1967	0·1089	—	0·0057	0·0452	0·0802	0	43·67	1	9·9
,, ,,	,,	1897	120	3·79	$\frac{7}{11}$ time water. $\frac{4}{11}$ time steam.	0·1473	0·0837	—	0·0053	0·0430	0·0424	0	32·17	1	4·1
Alaska Treadwell G. M. Co.	,,	1901	540	4	$\frac{3}{4}$ time water. $\frac{1}{4}$ time steam.	0·0974	0·0338	—	0·0026	0·0212	0·0336	3	18·59	0	9·3
Golden Star . .	Dakota . .	1894	160	4	Steam.	0·1556	0·1983	0·2043	0·0071	about 0·0210	0·1149	0	70·12	2	11·1
Homestake . .	,,	,,	80	4	,,	0·2513	0·2381	0·1986	0·0083	about 0·0210	0·1348	0	85·51	3	6·8
Plumas Eureka .	California.	1888	60	2·5	Water.	0·2400	-		0·0900	0·0850	0·0725	0	18·75	2	0·4
Bunker Hill . .	,,	,,	40	2·5	,,	0·1700		0·2000	0·0150	0·0650	0·1550	0	60·50	2	6·3
Mahoney . . .	,,	1896	40	3·5	,,	0·1230		0·0780	0·0070	0·0380	0·0720	0	31·80	1	3·9
North Star . . .	,,	1895	40	1·42	,,	0·1525		0·2609	0·0170	0·0650	0·0276	0	52·30	2	2·2
Wildman . . .	,,	,,	30	3·33	,,	0·1610	—	0·0910	0·0060	0·0310	0·0690	0	36·40	1	6·2
,,	,,	1888	10	2·25	,,	0·2600	-	0·1800	0·0100	0·0500	0·1000	0	60·00	2	6·0
Gover	,,	1893	20	2·2	,,	0·1600		0·2000	0·0100	0·0100	0·0700	0	48·00	2	0·0
Sutter Creek .	,,	1888	10	1·7	,,	0·3330	.	0·6670	0·0200	0·0800	0·0100	1	11·00	4	7·5
Loyal Lead . .	,,	,,	10	1·5	,,	0·6400	—	0·4000	0·0200	0·1400	0·2500	1	45·00	6	0·5
Esmeralda . . .	,,	,,	10	1·5	Steam.	0·5000	0·6700	0·0700	0·0200	0·1500	0·0900	1	50·00	6	3·0
Average of Californian Mills		1888	40	2·25	Water.	0·205	—		0·040	0·100	0·105	0	45·0	1	10·5
	Colorado	1871	—	0·75	Steam.	[illegible]	[illegible]		[illegible]	[illegible]		2	85	11	10·5

centration, here done on vanners. The additional cost occasioned by this operation was as follows in 1895:—

	$	
Labour	0·1006	per ton.
Supplies	0·0098	,,
Water (power)	0·0239	,,
Total	$0·1343	= 0s. 6·7d.

The milling of low-grade ore is a matter that has received much attention in the Western States; it may fairly be looked upon as the probable main cause that has induced American mill men to keep elaborate statistics of their operations, the study of which affords information of the greatest use in pointing out where any excessive outlay occurs, and in effecting corresponding economies. As a most instructive example of the detailed costs of all the items of large and thoroughly well-run mills, the full particulars of the Alaska Treadwell and Alaska Mexican mills, both situated at Douglas Island, Alaska, are here given for the year 1896. These mills, as already stated (p. 463), are working ores yielding about two-thirds of their values in the form of free gold and the rest as sulphurets. The results of the year's working were:—

	Alaska Treadwell.	Alaska Mexican.
	$	$
Free gold, per ton	2·00	1·67
Gold from sulphurets . .	0·97	0·75
Total value per ton . .	$2·97 = 12s. 4·5d.	$2·42 = 10s. 1d.
Percentage of sulphurets .	1·66	2·18
Costs of treatment per ton:—	$	$
Mining	0 54·91	1 14·11
Milling and concentrating	0 34·76	0 39·41
Chlorination	0 11·38	0 15·99
General expenses . . .	0 15·27	0 11·61
Total	$1 16·32 = 4s. 10·2d.	$1 81·12 = 7s. 6·6d.

The Alaska Treadwell mill consisted of 240 heads, whilst the Alaska Mexican mill ran 60 stamps only during some eight months and 120 during the remainder of the year; the former mill crushed 263,670 tons and the latter 101,702 tons.

The operating expenses are detailed as follows:—

		Alaska Treadwell.		Alaska Mexican.
Labour—		$ per ton.		$ per ton.
Foreman and assistant		0·0162		0·0297
Amalgamators		0·0192		0·0264
Feeders		0·0302		0·0285
Concentrators		0·0288		0·0327
Crushers		0·0142		0·0433
Carpenters		0·0117		0·0017
Ditchmen		0·0171		0·0039
Miscellaneous		0·0075		0·0058
Total labour		0·1449		0·1720
Supplies—	$ per ton.			$ per ton.
Battery shoes	(572—84,697 lbs.)	0·0240	(122—21,107 lbs.)	0·0147
Battery dies	(480—67,225 lbs.)	0·0173	(106—16,604 lbs.)	0·0117
Screens	(1300 sq. ft.)	0·0021	(310·5 sq. feet)	0·0016
Feeder extras		0·0021		0·0022
Crusher extras		0·0070		0·0139
Mortars	(2)	0·0018		}
Mortar liners	(58,058 lbs.)	0·0143		} 0·0123
Boss heads	(12)	0·0009		}
Cam shafts	(3)	0·0011		}
Lubricants		0·0015		0·0059
Concentrator extras		0·0115		0·0030
Mercury	(48 flasks)	0·0074	(14 flasks)	0·0056
Steam power account		0·0748		0·1195
Electric light account		0·0034		0·0102
Assay office account		0·0010		——
Repairs account		0·0170		——
Miscellaneous		0·0155		0·0215
Total supplies		0·2027		0·2221
Labour, as above		0·1449		0·1720
Total milling cost		0·3476 = 1*s.* 5·4*d.*		0·3941 = 1*s.* 7·7*d.*

In both of the above mines the milling costs are exceptionally low, owing to the large size of the mills, their favourable situation, and the fact that ample water power is available during a large part of the year. The advantage of this item appears from the table on page 474, whence it will be seen that in the year 1893 the cost of milling with steam at the Alaska Treadwell mill was 2*s*. 9d. per ton, whilst it fell to 1*s*. 5d. when running with water, the year's average cost being 1*s*. 10d. The chief economy that seems to have been introduced within the last few years is the replacement of the rockbreakers of the Blake type by Gates crushers; with the former the cost for labour was 0·4460 and of supplies $0·0143 per ton, these items with the latter machines being $0·0142 and $0·0070 respectively. The relative economy of a 120 head stamp mill as compared with a 60 head, is well illustrated by the figures for the Alaska Mexican mill given in the above table, whilst it will also be noted that the Alaska Treadwell mill of twice the size works very little cheaper. During the last seven or eight years, the cost of milling at the latter mill has only shown trifling, almost accidental, fluctuations, averaging about 1*s*. 6d. per ton. The cost of milling in the new mills is now as low as 18·59 cents. (say 9½d.) per ton; the wondeful economy here realised is due not only to the large size of the mills, but to their excellent equipment and careful management, the strictest attention being paid to every detail. Thus it is worth noting that the cost per ton of the steel shoes used is $0·0124, and of the dies, which are of cast iron, $0·0088.

With the above results may be compared those obtained in another very well-run mill, that of the El Callao Mining Company in Venezuela, which worked high-

grade ore under circumstances of special difficulty. The mill was a 60-stamp mill of good modern construction; during the year 1892, it crushed 52,823 tons, which yielded at the rate of 12 dwt. per ton.

The costs per ton, the mill crushing at the rate of 2·6 tons per stamp head per 24 hours, were as follows:

	s.	*d.*
Labour	1	7·2
Wood	1	5·1
Mercury	0	0·8
Shoes and dies	0	4·4
Repairs	0	9·9
Mill supplies and miscellaneous expenses	0	8·3
Total milling cost	4	11·7

In Mexico the Mesquital del Oro mine runs a 50-head mill of eight (650 lb.) stamps, crushing at the rate of 2·2 tons per head per 24 hours; the fuel used is wood which costs 15*s.* per cord of 121 cubic feet, the mill using 11 cords per 24 hours. The costs per ton in 1893 were as follows:

	s.	*d.*
Wages (white men)	0	10·75
,, (native)	1	1·25
Supplies	1	4
Firewood	1	5·75
Total	4	9·75

The cost of milling at the Spanish mine, Nevada County, California (already referred to, page 303), is worth recording as an example of a mill doing exceptionally good work under highly favourable conditions. The ore is soft and easily reduced, consisting of a soft talcose slate with stringers of quartz. It is crushed in four Huntington mills, whose crushing capacity is somewhat over 4,000 tons per month, or say

140 tons per day. The costs per ton were as follows about 1894:

	$
Labour	0·110
Mercury	0·005
Shoes and dies	0·046
Mill supplies	0·037
Miscellaneous	0·015
Water	0·036
Total milling cost	$0·249 = 1*s.* ·05*d.*

The ore is exceedingly low grade, yielding only about 3*s.* 3*d.* per ton; but even at this low figure the operations still result in a small profit, the total cost of mining and milling being about 2*s.* 2*d.* per ton. The extraction is poor, being under 50 per cent. of the value of the ore by assay.

In contrast to this type of work, the cost of running a good arrastra may be quoted. Such is a custom arrastra running on high grade ores in the Carson district. The ore is broken to the size of hazel nuts before being charged into the arrastras, of which there are two; each can work off a charge of about 7 cwt. of ore in 6 to 8 hours, the average capacity of the entire plant being 5·4 tons in 24 hours. The power used is water during eight months of the year, and steam during the remaining four. The charge for treating ores varies with the size of the parcel, the minimum rate being $15 per ton. The following is the approximate cost of treating one ton of ore:—

	$
Labour	4·18
Fuel	2·35
Mercury	0·81
Supplies	0·79
Total cost	$8·13 = £1 13*s.* 11*d.*

Such high costs are of course only permissible when small quantities of high grade ore presenting special difficulties have to be treated.

Arrastras have, however, the additional advantage that they can be operated by animal power as well as by steam or water, are good gold savers, and are cheaply erected. A pair of first class arrastras 12 feet in diameter, driven by a wooden hurdy-gurdy wheel 13 feet in diameter, cost in California in the year 1895, complete in running order, \$700 or about £145. It takes 30 miner's inches of water under 100 feet head to drive it; one man, on a 12-hour shift can attend to it, and it treats about 5 tons in 24 hours, each arrastra treating a charge of $1\frac{1}{4}$ tons of ore in 12 hours.

Nova Scotia.—A few figures are available showing the cost of milling in Nova Scotia, and are presented in tabular form on page 482. Fuel and labour are both cheap, hence the costs are low, having regard to the small size of the mills. These figures again refer to the short ton of 2,000 lbs.

It may be noted that at Sherbrooke the charge for custom milling is \$1·15 (4*s.* 9*d.*) per ton. As these figures will show, the conditions in Nova Scotia are favourable to extremely cheap milling, and if the gold-mining industry develops as it should do, so as to permit of the erection of larger mills, this colony will be able to rival most other countries in its treatment of low-grade quartz. It must, however, be remarked that most of the reefs hitherto worked are comparatively narrow, hence the cost of mining is proportionately higher.

Australia.—In Australia a large proportion of all the

mills are custom mills, whose interest it is of course that the exact cost to them of crushing quartz shall not be ascertained. Partly no doubt for this reason, and partly perhaps because gold milling has not become so closely studied a science in the older gold-producing colonies as in other countries which have to treat poorer ores, there are but meagre data available as to the cost of milling. Gold milling in Western Australia, again, is in a more or less experimental stage, so that the real cost there of milling may be said to be almost unknown up to the present. A few data as to costs are shown in the table on p. 483.

This table shows very extreme variations of milling cost, and would alone suffice to indicate that stamp-milling in Australasia is still very far from having advanced to the dignity of an exact science. According to an official communication, the cost in the different districts of New South Wales varies from 2*s*. 6*d*. to 6*s*. 6*d*. per ton. Some of the very best equipped and managed mills in the older colonies, and especially in New Zealand, are working at costs not far exceeding 3*s*. per ton, whilst in the majority the cost is between twice and thrice that figure. It is almost as rare to find a mill in these colonies crushing for less than 5*s*. per ton, as it is to find a Californian mill in which the cost exceeds this figure. In Queensland a 30 head Customs mill charges 12*s*. per ton for crushing, including cartage of the quartz for $1\frac{1}{2}$ miles from mine to battery. Labour is dear, and fuel (wood) costs about 21*s*. per cord. In West Australia, where fuel and water are both costly items, milling seems to cost about 10*s*. per ton.

India.—The following are the costs of milling on two

Nova Scotia.

Name of Mine	Date.	Number of Stamps in operation.	Tons crushed per Stamp per 24 hours.	Nature of Motive Power.	Cost per Ton crushed						
					Labour.	Fuel.	Mercury.	Shoes and Dies.	Mill Supplies.	Total.	
										Canadian Currency.	Sterling.
					$	$	$	$	$	$ cts.	s. d.
Oldham (on custom work) .	1893	10	2·9	Water.	0·2730	—	0·0205	0·0264	0·0293	0 34·92	1 5·5
Oldham (working continuously) .	,,	do.	do.	,,	0·2100	—	0·0205	0·0264	0·0293	0 28·62	1 2·3
Waverley . . .	,,	10	?	Steam.	—	—	—	—	—	0 48·10	2 0·1
Oxford	1885	10	0·5	,,	0·7600	0·3200	—	0·1500	—	1 23·00	5 2·5
Sherbrooke . . .	—	15	1·0	Water.	0·2700	—	0·0200	0·0300	—	0 32·00	1 4·0
Brookfield . . .	1898	20	2·0	Steam.	0·3760	0·1480	0·0080	0·0670	0·0260	0 62·50	2 7·3
Richardson . . .	1898	40	2·45	Steam.	0·2140	0·0800	0·0090	0·0230	0·0290	0 35·50	1 5·8

Name of Mine.	District.	Colony.	Date.	Number of Stamps.	Tons crushed per head per 24 hours.	Nature of Motive Power.	Cost per (long) ton crushed.							
							Labour.	Fuel.	Water.	Shoes and Dies.	Supplies.	Miscellaneous.	Total.	
							d.	d.	d.	d.	d.	d.	s.	d.
Wimmera	Stawell.	Victoria.	1898	55	1·55	Steam	30·0	12·0	1·0	1·9	31·1		6	4·0
Magdala cum Moonlight . . .	,,	,,	,,	30	2·0	,,	41·66	9·30	0·30	4·50	8·00	11·24	6	3 00
New Moon . .	Bendigo.	,,	,,	—	—	,,	139·50	5·25	—	27·75		16·00	15	8·50
South Clunes United	S. Clunes.	,,	1892	60	2·5	,,	10·5	—	—	3·3	—	—	2	3·0
Victory	Charters Towers	Queensland.	1898	15	—	,,	262·8	74·9	—	96·5		2·3	36	4·5
New Queen	,,	,,	,,	25	2	,,	69·0	23·9	—	13·3		9 5	9	7·7
Brilliant Block . . .	,,	,,	,,	40	1·4	,,	37·6	20·7	—	12·4		11·4	6	10·1
Day Dawn P. C. . .	,,	,,	,,	50	—	,,	97·3	46·4	—	21·1		2·4	13	11·2
Saxon	Hauraki.	New Zealand.	1892	33	1·9	Water	13·0	—	4·0	—	—	—	4	1·0
Te Aroha	,,	,,	,,	60	1·5	,,	33·0	—	—	—	4·0	—	3	1·0
Woodstock	Karangahaki.	,,	1898	30	1·2	,,	—	—	—	—	—	—	12	7·5
New Zealand Crown	Auckland.	,,	,,	60	1·6	,,	—	—	—	6·25	—	—	3	3·0
Great Boulder Proprietary	Coolgardie.	West Australia.	1897	30	2·8	Steam	64·2	14·4	26·0	15·9		7·0	10	7·5
Menzies Consolidated	,,	,,	,,	20	—	,,	—	—	—	—	—	—	12	10·9
Golden Horseshoe .	,,	,,	1901	50	4·729	,,	19·68	22·74	29·83	2·13	41·31		9	7·69
Ivanhoe Gold Corporation	,,	,,	,,	40	3·97	,,	22·34	21·61	21·58	9·92			6	3·45

of the most successful mining properties of the Mysore gold fields in 1897, the ton being here the long ton of 2,240 lbs. :—

		Mysore.		Nundydroog.	
Number of heads of Stamps		120		70	
Tons per head per 24 hours		2·1		2·4	
Tons crushed per annum		74,272		49,675	
		s.	d.	s.	d.
Cost per ton.	European labour	0	7·97	0	7·07
	Native labour	1	1·36	0	10·24
	Fuel	3	1·22	2	0·25
	Shoes and Dies	0	2·49	0	2·74
	Mercury	0	0·48	0	0·57
	Lubricants	0	1·39	0	1·21
	General Stores	0	5·52	0	9·00
Total		5	8·43	4	7·08

It will be noticed that one half of the milling cost represents the cost of fuel, due mainly to the fact that coal at the mines is worth some 30*s*. per ton. The costs at these mines have been very considerably reduced in recent years owing to the increased size and improvements in working of the mills, and largely to the modernisation of the process employed. Thus in 1901 the Mysore mill crushed 127,070 tons at a cost of 4*s*. 3·9*d*. per ton, and the Nundydroog 52,030 tons at a cost of 4*s*. 2·7*d*. per ton. These mines used formerly to re-treat the whole of the tailings from the stamp-mills by the antiquated system of pan amalgamation, the cost of which in 1897, when it was still in use for a portion of the mill tailings, is shown in the following table :—

		Mysore.		Nundydroog.	
Quantity of tailings treated.		47,895 tons.		8,010 tons.	
		s.	d.	s.	d.
Cost per ton.	European labour	0	8·19	—	—
	Native labour	1	2·24	1	10·12
	Fuel	3	3·25	3	4·10
	Shoes and Dies	1	8·31	0	10·94
	Mercury	0	3·59	0	0·94
	Lubricants	0	1·97	0	2·70
	General Stores	0	6·70	0	2·59
	Total	7	10·25	6	7·39

The process is not only costly, but also inefficient, the tailings from the pans requiring apparently to be again treated by cyanidation. The cost of this latter process in the above mines worked out at 3*s.* 0·7*d.* and 2*s.* 3·7*d.* per ton treated respectively in 1901. It is therefore evident that the proper method of working here, as elsewhere with similar ores, is by stamp mill amalgamation followed by cyanidation; and this is the process that has now been substituted for the old one by these companies. A neighbouring company, the Champion Reef, working 140 heads of stamps, obtained the following results in 1898:—

	Tons treated.	Cost per ton.	
		s.	d.
Stamp Mill	89,271	6	3·64
Pan amalgamation	28,761	7	0·96
Cyanidation	73,202	4	7·22

Yet another, the Oongaum mine, crushed 78,125 tons in 1901, the costs per ton being as follows:—

	s.	d.
Labour	1	9·3
Fuel	3	2·5
Shoes and dies	0	3·5
General stores	0	7·7
Total	5	11·0

At these mines the cost of cyanidation amounted to 3*s*. 0·9*d*. per ton, out of which one-half was for cyanide.

In all these mines native labour is of course cheap. The yield of gold varies from about 1 ounce to rather over 1½ ounces to the ton of stone crushed.

Great Britain.—The cost of milling at one of the very few British Gold mines, where systematic working on a considerable scale has ever been attempted, the Morgan mine in Wales, is a matter of considerable interest. It was as follows at the beginning of 1893:—

Average number of heads running	40
Total time of running	504 hours.
Number of tons crushed	1,150
Tons crushed per head per 24 hours	1·37
Motive power	water.
	s. *d.*
Cost per ton—labour	0 10
,, ,, ,, material	0 1¼
Total	0 11¼

In the years 1894–1895, the total quantity milled was 12,354 tons, at an average cost of 2*s*. 3·40*d*. per ton; the latter was very irregular, the monthly averages fluctuating between 1*s*. 4·96*d*. and 4*s*. 5·06*d*.

On the Continent of Europe there are but few examples of gold milling, properly speaking. At Vulkoj in Siebenbürgen, Transylvania, a 20-head stamp mill crushes at the rate of 1·75 tons per 24 hours; it is worked by steam, and the sulphurets are concentrated on Frue vanners. The cost of milling is returned at 6*s*. per ton. The largest stamp-mill in Europe is said to be at Gurabarza, near Brad, Transylvania, containing 190 stamps.

At Bömmel, in Norway, a 50 stamp-mill was in operation till recently, with a crushing capacity of 1·34 tons per

head, per 24 hours; in 1896 it cost 21·3*d.* for fuel and 5·3*d.* for wages per ton crushed.

Transvaal.—The costs of milling at a few typical mines are given in the table on page 488.

As a rule the best mines on the Witwatersrand mill at a cost of between 2*s.* and 2*s.* 6*d.* per ton, very few exceeding 3*s.* That so low a cost can be attained in spite of dear fuel, and the high costs of stores and wages, is due to the large size of the mills, their very perfect construction, economical management, and also to the fact that coarse crushing is the rule, the fine gold being extracted from the mill tailings by cyanidation. At the Sheba mine the cost is higher, in spite of the existence of available water power, because a system of much finer crushing is employed; whether this is an economically sound system is perhaps open to doubt. During 1898 one-sixth of the total ore treated at the Sheba was crushed by the use of electricity, and five-sixths with steam (using coal at 17*s.* 6*d.* per ton as fuel) for the motive power. In the previous year only sixty stamps were run entirely by electricity, and the cost of this power per ton crushed was 2*s.* 3·376*d.* as against 1*s.* 1·233*d.* given above for steam and electricity. The central mill of the Transvaal Gold Mining Estates, Limited, is entirely driven by electricity; its costs would be lower than they are were it not for the high price of all stores in this somewhat remote region. During the years 1892 to 1895 the cost of crushing in a 20-stamp-mill at the same place, averaged about 4*s.* per ton.

Labour.—It has become a custom almost universally established in mining centres that mill-men work 12 hour shifts. This indeed they can easily do, as the labour is but light, except when anything goes wrong; then all hands have to work as hard as they well can in

TRANSVAAL.

Name of mine.	District.	Date of statement.	No. of stamps.	Tons crushed per head per 24 hours.	Costs per ton of 2,000 lbs.								
					European labour.	Native labour.	Fuel.	Mercury.	Mill spares.	General supplies.	Miscellaneous (including maintenance).	Total.	
					d.	d.	d.	d.	d.	d.	d.	s.	d.
Crown Reef G. M. Co., Ltd.	Witwatersrand	1898	120	4·608	9·17	2·14	7·90	0·43	4·50	5·80	5·35	2	11·29
Geldenhuis Estate & G. M. Co., Ltd.	,,	1898	120	4·71	6·48	4·01	5·59	0·33	3·20	1·91	1·37	1	10·89
Langlaagte Estate & G. M. Co., Ltd.	,,	1897	200	4·48	5·36	2·25	7·14	—	2·59	2·12	9·98	2	5·44
The Jumpers G. M. Co., Ltd.	,,	1898	100	4·08	8·97	10·85	11·28	0·65	5·93		12·83	4	2·51
Sheba G. M. Co., Ltd.	Barberton	1898	120	2·856	26·311		13·233	27·103			26·099	7	8·746
Transvaal G. M. Estates, Ltd.	Lydenburg	1898	60	3	—	—	—	—	—	—	—	2	4·88

order that the least time possible may be lost. Whether the rock-breakers be in a separate rock-breaker house, or whether they occupy their floor in the mill-building, the men who attend to them only work a 10-hour shift, rock-breakers never being run during the night shift except in cases of emergency. The capacity of the rock-breaker plant should always be so calculated as to be able to crush easily in 9 hours as much as the mill can do in 24, and if there are ample storage-bins for the crushed ore after it has left the rock-breaker, the rock-breaker plant may be even more powerful with advantage. One man can readily attend to a rock-breaker, help in dumping the ore cars as they arrive, keep the grizzlies clear, and do all the necessary work, including the keeping of a tally of the number of cars handled, and in case of rotary crushers like the Gates or Comet, which requires practically no attention, one man can even attend to several crushers.

A small mill, say of 20 stamps, requires a staff of two amalgamators, one on each shift, and two concentrator men. The former attend to the battery, repair the screens, make wedges for the stamp-heads, and do other light work. Larger repairs involve the assistance of a mechanic or a carpenter. The concentrator men see to the proper running of their machines, adjust the water supply, inclination of the tables, &c., and wheel out the concentrates from time to time. Of course, it is understood that both feeding and concentrating are done by machinery. When hand-feeding is employed, one man is required on each shift to every 20 heads, and when blanket-strakes or hand-buddles are used, a boy, or even two, will be required to assist the concentrator man. In addition to the above, assaying, retorting, melting, and

general supervision will require part of the time of the mine superintendent or manager, who should be capable of discharging all these duties.

Coming to larger mills, practically the same staff would work a 40 stamp mill as a 20 stamp, namely :—

1 man at rock-breaker,	working	10	hours.
2 amalgamators . . .	,,	12	,,
2 concentrator men .	,,	12	,,
1 labourer	,,	10	,,
6 men per day of 24 hours.			

At the Homestake and Golden Star Mills in Dakota, of 100 and 160 stamps respectively, the following is the staff of men employed :—

	Homestake.	Golden Star.
Amalgamators	5	5
Rock-breaker men	2	2
Millwright	1	1
Oilers	2	2
Feeders	2	4
Labourers	2	4
	14	18

To which must be added part of the time of foremen, carpenters, machinists, and watchmen, about equivalent to the work of an additional man, thus bringing the above numbers up to 15 and 19 respectively. These being steam mills, the labour at the boilers has also to be included, this being in each case 2 enginemen and 2 firemen, or an addition of 4 men to each staff, making the total number of hands employed about 19 and 23 respectively.

In the well arranged mills of the Alaska-Treadwell Company (240 stamps) and of the Alaska Mexican Company (120 stamps), the respective labour staffs average about as follows :—

	Alaska—Treadwell.	Alaska—Mexican.
Foreman and assistant	2	2
Feeders	8	4
Concentrator-men	8	4
Amalgamators	4	2
Crusher-men	3	6
Oilers	2	—
Ditchmen	3	1
Carpenters	2	1
	32	20

In each case, a little additional time from machinists, &c., is required, so that the labour force of the two mills together may be taken at about 53 per diem or about 600 hours; the total quartz that the two mills are capable of crushing is about 1190 tons daily, so that roughly each ton of quartz crushed requires the labour of one man for 0·5 hour.

At a large Transvaal mill, the New Primrose, with 160 stamps, crushing 820 tons of ore per day, the labour staff is about as follows:—

Battery manager and mill-foremen	3
Amalgamators	9
Mechanic and carpenter	2
Engine drivers and stokers	6
European labour	18
Natives	28
Total	46

On account of the high crushing capacity of the stamps, the labour ratio here is low, amounting to 0·23 hour of European labour and 0·34 hour of native labour per ton of ore crushed, these figures excluding both rock-breaking and concentrating. It is, however, scarcely possible to compare the labour efficiency of a native and that of a white man about a stamp-mill.

Taking the above (p. 490) staff for a 40-stamp mill, capable of crushing 120 to 160 tons per day, the labour equivalent of one man working about 90 hours when water-power, or 140 hours when steam-power is employed, will be required per day; this will give as the labour equivalents for one ton of quartz crushed 0·7 and 1·0 hour respectively of one man's work. When the average rate of wages in a district is known, these figures give fairly reliable data for estimating the approximate cost of labour incurred in milling a ton of quartz in a mill of the above size.

Cost of Mills.—The first cost of a stamp mill varies considerably according to the quality of the machine and the items included in the bill of charges. As a rough estimate it may be taken that the cost of a mill from rock-breaker to mercury traps, but excluding motive power or mill buildings, is about £100 per head, and of the bare mill alone about £50 per head, a little more for mills of less than 15 heads, and a little less for mills of over 40 heads. A first-class English-built mill of 60 nine hundred pound stamps with knee frames complete, including Hendy's challenge feeders and good rock-breakers, cost £6,500, this being exclusive of motive power. A 20-stamp mill of seven hundred pound stamps, complete with galvanised iron building, and engine and boilers, cost £3,900. In California a 20-stamp mill with nine hundred pound stamps, complete with Pelton wheel, ore bins, mill-house, &c., was erected by contract for $12,800 (say £2,600). A first-class 40-stamp mill complete with Pelton wheel, but without mill building, cost $22,000 (say £4,400) in 1892. In 1896 another 40-stamp mill complete, ready to run, was erected in California for $21,000 (say £4,200). A 60-stamp mill of 1020 pound

stamps for Alaska cost \$40,600 (say £8,200), freight, erection, buildings, &c., bringing the price in running order up to \$60,000 (say £12,000). The new 300-stamp mill seems to have cost (including the mill, site, &c., about \$320,000 (say £65,000). In Australia a 10-stamp mill with cast-iron frames, exclusive of everything except the mill and frames, costs about £450, this being about the same price as would be charged in England.

The costs of some of the larger mills in the Transvaal are given in the following table, based upon the published accounts of the companies in question:—

	No. of stamps.	Total cost. £	Cost per head. £
New Primrose	160	90,000	562
Geldenhuis Estate .	120	46,000	383
Crown Reef	120	59,000	492
Glencairn Main Reef .	160	94,000	588
Block B, Langlaagte .	80	33,000	412

The above figures include mostly the mills erected and in complete running order, with buildings, shops, engines, boilers, water supply, &c., but mostly without rock-breakers and in some cases without concentrators. It has been estimated that machinery erected on the Witwatersrand cost two and a half times as much as its price in England, and this estimate is not far off the mark, though perhaps a little low for the stamp mills quoted above; three times the home price would seem to be a fairer figure, though very much must naturally depend upon the outlay required to get a sufficiently good foundation.

Specifications.—Care should be taken that these are full and inclusive; attention is here only drawn to a few special points, whilst by way of example a full specification for a 20 head mill, for which I am indebted to Messrs. Fraser and Chalmers, Limited, is given in appendix B, page 578.

Mortars.—Their approximate weights should be specified; they must be planed on bottom and on screen seats. Each should be supplied with its set of steel liner plates, generally five in number, weighing together about 4 cwt. Chock-blocks and screen frames should be supplied, and screen keys of forged steel. The hold-down bolts should be 1½ inches in diameter.

Stamps.—The stem should be of best hammered scrap or forged Bessemer steel. The heads may be cast iron or steel. The tappets should be of cast iron or steel, bored to gauge and fitted with steel gibs and keys. The shoes should be of chrome or manganese steel, and the dies of softer cast or forged steel, with a due supply of spare ones. Drifts for driving out the stems and shoes should be supplied.

Cam-Shafts should be of Bessemer steel turned to gauge. Cams should be of good cast steel, planed on the inner face and polished on the working face. The bearings should be of cast iron and well babited. The pulleys should be of wood with cast-iron bosses.

Guides.—Adjustable guides should be employed.

Frames.—Their nature, the quality of their timber, and dimensions of the principal beams should be clearly stated. They must be painted or tarred. The battery blocks must be of suitable length and quality.

If ore bins are required, ore bin gates should be included; if Frue vanners, proper launders for distributing the pulp; if rock-breakers, grizzlies must be included.

A proper sized water main and supply tanks should be furnished.

The length and gauge of copper tables should be stated. The requisite clean-up gear, say one barrel,

one batea, and one clean-up pan for each forty stamps will be required; also retort with condenser, and mercury-room fittings. Motive power should be taken well in excess of the apparent requirements of the mill. All the main shafting and counter-shafts should be of ample strength.

Mill Sites.—Economy in working a mill depends to some extent upon its situation. This is generally determined by its requirements in the form of water. The advantages of using this form of power are so great that it is very often worth while to construct a tram-line of considerable length, whenever by so doing water-power can be utilised; hence the majority of mills are situated on the banks of streams or rivers. Of course, the water-motor will need to be as low down in the valley as possible, in order to utilise the full available fall of water the more effectually. Care, however, must be taken to keep the mill well above flood level. If the waste tailings cannot, for any reason, be carried away by the stream, the mill must be at such a height above the bottom of the valley as to leave ample room for tailings' dumps, and this is even more important when working an ore of such high grade that the tailings are worth saving for re-treatment, in which case due provision must be made for the necessary retaining dams or reservoirs. Whether the mill is worked by water or by steam, proper arrangements must be made for an ample supply of water to the battery. If possible this should be brought in by gravity, but where this is impossible, pumps of sufficient capacity, not only to provide the normal quantity used, but to meet any emergency, should be laid down. However the water is obtained, it should be carried into reservoirs or tanks at

such a height above the mill floor as to give at least a 20 to 30 feet head. These reservoirs should hold a six hours' supply for the entire mill. They should in case of need be preceded by settling tanks, or even by rough filters if the water is muddy or foul from any cause, seeing how serious may be the losses of gold caused by using muddy water in the battery. If at all possible, a supply of good clean water should be obtained for the mill, as much of the success of amalgamation depends upon it. In cold climates provision must be made for heating the water supplied to the mill. This is best done by means of steam led into the tanks from a separate boiler, built especially for this purpose; waste steam from the engines must on no account be used, as it is sure to carry some grease with it. In Nova Scotia, the custom mill at Oldham, which is run by water-power, makes an additional charge of $3\frac{1}{2}$ cents of a dollar in winter time for the fuel required to heat the battery water.

It occasionally happens that mills have to be erected in places where the water supply is insufficient for the needs of the battery. When this is the case, tailing reservoirs have to be provided, in which the tailings and slimes are allowed to settle as completely as possible, the water being then pumped back to be used over again. This is done by a well-planned arrangement at the Highland Mill, Dakota; two reservoirs are formed, one above the other, by throwing two dams at different levels across the valley of a small creek, the lower reservoir thus formed being four times the size of the upper one. The tailings are run into the upper reservoir, where the sands are allowed to settle, the slimes overflowing from it into the lower one; the slimes settle there, and clear

water is pumped back from it into the mill supply tank. An arrangement is provided by which these reservoirs can be sluiced out when they are filled with accumulated tailings. In case of need, mine water can be used for supplying mills, but as this is mostly muddy, provision has to be made to allow the suspended matter in it to settle. Water from some mines, especially from those where the ore is rich in pyrites, is apt to be acid; when this is the case, sufficient lime should be thrown into the settling tank from time to time to neutralise such acidity and to precipitate any iron that may be present.

When the location of a mill is not determined by its proximity to water, there are a few other circumstances that influence its selection. Thus it is always advisable, when possible, to choose the side of the hill having a fairly rapid slope, and consisting of a rock sufficiently solid to form a good foundation for the battery. The average height of a mill of the "high" type, with a rock-breaker floor above ore bins of normal capacity, may be taken as 50 to 55 feet from the level of the car track to that of the concentrator floor, the length over all of the profile of the mill being rather over 100 feet. Hence a slope of 1 in 2 is the most suitable for a mill site. Of course provision must be made for the bringing in of a good track from the mine for the ore cars, and for the easy transport of fuel, where it is required, to the boilers. Some mills have been very successfully supplied with ore by means of wire-rope tramways, which automatically discharge into the proper ore bins. This arrangement is specially advantageous when a comparatively short distance has to be traversed in very hilly country.

With the modern methods of transmitting power

electrically, the close proximity of the mill to a source of water power is becoming of less importance, and on account of the readiness with which power so conveyed can be subdivided most of the objections to a separate crusher house disappear. It is therefore becoming less important to locate mills on the sides of steep valleys, and any site with a sufficient slope to carry away tailings is considered satisfactory provided that it offers good solid foundations. Under these conditions a site close to the principal mouth of the mine, or equidistant from all of them, if there are several, is apt to be preferred; the great principle must always be remembered that, for economic working, the ore should never be lifted from the time it enters the battery until it escapes after final treatment, in the form of exhausted tailings, but should continuously descend from stage to stage moved by gravity alone. When tailings have to be submitted to cyanidation after leaving the mill, it is rarely easy to get a site such that the tailings can be carried down to the vats by gravity, and it is usual to elevate the tailings from below the mill to the top of the cyanide plant. This may be done in various ways: bucket elevators or tailing wheels may be used, which are quite automatic, or else the tailings may be filled into cars, which may be hauled up inclines or hoisted up vertically.

Mill Building.—A mill should always be enclosed in a substantial building, admission to which can only be obtained at one or two points. In America these structures are mostly all wood, but this material, though possessing some advantages, presents great risk on account of fire. An iron building is better, but one with

brick walls and a galvanized iron roof is better still. The system prevalent in America of building all the roof in one plane is also not to be recommended, as such a roof is especially liable to damage from storms. It is better built in three or four bays, which moreover give excellent opportunities for lighting and ventilation. The roof should be carried entirely upon the walls of the building, or else upon independent wrought-iron columns; it should not depend for support on any portion of the mill framing, which is certain to be affected more or less by the vibration of the stamps. It is also a good plan in large mills to support the rails upon which the crawls used in hoisting the stamps run, by means of the same pillars that carry the roof, instead of fastening these to the battery uprights. It is well to bear in mind in designing a mill building that these pillars must, if possible, be so placed as not to prevent the mill man from getting an uninterrupted view of the tables from any part of the mill. It may be taken that the average price of a suitable galvanized roof with pillars, &c., for a big mill, will be between 1*s*. 6*d*. and 2*s*. per square foot, and its weight about 1 cwt. for each 10 square feet of superficial area covered.

Power.—It has already been pointed out how the power required to drive any given stamp mill may be calculated. From this, and knowing the approximate power required to work the various other machines in the mill, the total power required may readily be estimated. Thus the power required to work a 40 head battery of 900 lb. stamps, designed to make 90 seven-inch drops per minute, and having a capacity of 140 tons of quartz per 24 hours, will be as follows :—

Rock-breaker to crush 15 tons per hour	30	I.H.P.
40 stamps at 1·7 I.H.P. each	68	,,
16 Frue vanners	10	,,
1 amalgamating barrel	2	,,
1 batea	1	,,
1 clean up pan	2	,,
	113	,,
Add 5 per cent. for friction of shafting	6	,,
Total power required	119	,,

This makes the efficiency of the stamp mill about $1\frac{1}{6}$ ton per I.H.P. per 24 hours including amalgamating plant.

This is, however, scarcely a correct way of estimating the efficiency of a mill, the only really satisfactory way of estimating the effectiveness of any given machine being to calculate the quantity of quartz crushed per I.H.P. per hour, exclusive of the power required for after treatment of the amalgam, &c.

Thus for the above 40-stamp mill the amount of work done would be:

By the stamps 24×68	= 1,632	hours at 1 I.H.P.
By the rock-breaker 10×30	= 300	,, ,,
Total for crushing 2,800 cwt.	= 1,932	,, ,,

thus giving $\frac{2,800}{1,932} = 1{\cdot}45$ cwt. per I.H.P. per hour. To these figures 5 per cent. may be added as above for friction of shafting, when we should have 2,800 cwt. worked for 2,029 I.H.P. hours, or 1·38 cwt. per I.H.P. per hour.

It has already been pointed out that the effective crushing power of the Krom roll is about 1·04 cwt. per I.H.P. per hour. For a Huntington mill including rock-breaker it would be 1·0 cwt. per I.H.P. per hour (a five-foot mill crushes 20 tons in 24 hours using 12 I.H.P. + rock-breaker 12 H.P. during 10 hours).

The stamp mill thus seems to be the most efficient machine, even taking the figures as here given; these are, however, exceeded in some cases, seeing that in Dakota $4\frac{1}{2}$ tons, and in the Transvaal over 5 tons per head per 24 hours are sometimes crushed with stamps similar to the above.

Water Power.—It has already been pointed out that the best way of supplying motive power is by water, whenever possible. There are various forms of motors in use, according to the nature of the water power available. For high falls, the best motor is undoubtedly the Pelton wheel or some wheel of similar type, such as the Knight, these being actuated by a stream of water impinging from a nozzle against buckets attached to the periphery of a strong wheel. For falls of 100 feet, or more, this is certainly the best type of motor, and it may be used with advantage for falls as low as 50 feet. This wheel has the great advantages of cheapness and ease in erection, and a very high degree of efficiency is claimed for it; it will perhaps be safe to assume its factor of efficiency at 0·75 in practical working. For falls of less than 50 feet the best motor is a Jonval turbine, and where considerable power is required, a twin Jonval turbine, consisting of two wheels keyed upon one horizontal shaft, with a central supply pipe and each discharging in opposite directions, so that the entire machine is completely balanced, presents many advantages. Not the least of these is that a portion of the fall can be utilised beneath the turbine by the employment of a suction tube, which must not, however, exceed about 15 feet in length, so that the turbine itself can be placed clear of floods as a general rule. If, however, the water supply is subject to excessive variations of volume, a Girard turbine will be found superior

to the Jonval; the former turbine will continue to work when completely submerged in water, and retains practically its full efficiency when working with a water supply even very greatly reduced below the normal quantity. Turbines can be successfully employed when a fall of not more than 5 feet is available, provided that there is a sufficient volume of water to produce the power required. Broadly speaking, the efficiency of a well-constructed turbine may be put down at 0·65, or somewhat less in the case of very low falls. Another form of motor sometimes used for driving small mills is the Vortex wheel, which is, in fact, only another form of turbine, rather less efficient than the two first-named but cheaper as to first cost. In order to determine what power can be obtained from any given water supply, the total effective height and quantity of water available must be known, due allowance being made for the loss of head caused by friction in the pipes, &c.

Let n represent the number of cubic feet of water available per minute; h the net available height after making all deductions:

Then the theoretical horse power developed $= \dfrac{n\,h\,62{\cdot}5}{33{,}000}$

and the actual horse power obtainable $= \dfrac{n\,h\,62{\cdot}5}{33{,}000} \times$ factor of efficiency of the motor selected.

In this connection it is well to remember that a miner's inch of water in California is about 1·6 cubic feet per minute, and that a Government sluice head in Australia is 60 cubic feet per minute. Table I. gives, in a convenient form, the values of the fraction $\dfrac{n\,h\,62{\cdot}5}{33{,}000}$ worked out for a number of different values of n and h.

Electrical Transmission.—Of recent years great strides have been made in the application of electricity to the transmission of power to considerable distances, and mining engineers have not been slow in availing themselves of the advantage of this novel method. By its means mills, a good many miles away from water power, can yet be worked by it, the only additional expense incurred in running being interest on the cost of the transmission plant, and the wages of a competent man to look after it. Even with these additions it costs far less than does steam power, and has been applied with great success, notably in the Western States of America, the Transvaal, and in New Zealand.

The study of the details of any particular case of electric transmission is obviously a matter for the electrician and not for the mill man, so that only a few general hints on the subject will be given here; for most of these I am indebted to Mr. W. M. Thornton.

The various systems of electrical transmission now considered practicable are as follows:—

1. Continuous current.

2. Alternating current, either one-, two- or three-phase; using with either of the two latter either synchronous or asynchronous motors.

Continuous current is not well suited to distances of over 1,000 yards, and is best restricted to half that distance; the plant is simple in character, cheaper than for alternating currents, and is well adapted for minute subdivision and for lighting purposes. The motors will start at practically full load and are highly efficient; hitherto, however, it has been found difficult to construct direct current dynamos to work at high potentials, so that this system can hardly be employed where any

large amount of power has to be transmitted. In spite, therefore, of some very evident advantages possessed by the continuous current system, it is practically only used where comparatively low powers have to be transmitted for short distances.

This will rarely be the case when stamp mills have to be electrically driven, so that most modern installations employ alternating currents; electricians are, however, by no means agreed which form of alternating current is the most suitable under any given set of conditions. The three-phase system requires a smaller consumption of copper for the conductor than do any of the others, as shown by the following table of the ratios of the amount of copper in a line required to transmit equal currents for the same type of motor:—

Single phase.	Two wires.	Ratio of copper required	1
Two ,,	Four ,,	,, ,, ,,	1
Two ,,	Three ,,	,, ,, ,,	0·85
Three ,,	Three ,,	,, ,, ,,	0·75

The two-phase motor is moreover more difficult to start than the three-phase against a full load. On the other hand, two-phase transmission seems to be preferred by some authorities when the current has to be much subdivided and to be used for lighting.

In all systems the voltage should be kept high for the sake of economy in the line wire. With alternate currents high potential transmission is easily obtained even with relatively low potentials in the dynamo and motor, by the use of stationary step-up and step-down transformers. The cross section of the conductor to be employed can be roughly arrived at by the use of Lord Kelvin's law that "the interest on the first cost of the conductor plus allowance for depreciation should equal the cost of the

energy annually dissipated in the conductor." The voltage of a current equal to 1,000 ampères per square inch will drop 86 volts per mile, the loss varying directly with the number of ampères per square inch and with the distance. The conductors used in practice are usually between a No. 10 wire for 13 ampères and a 37/15 stranded, for 155 ampères ; tables showing the conductors to be used for any given current and loss of voltage are issued by all makers of electric cables. Lines are best carried on poles 25 to 30 feet high, of wood or iron according to circumstances, with porcelain insulators ; in wet climates oil insulators may be used. The number of poles may vary from 30 to 60 per mile. In most countries it is found necessary to protect the line by efficient lightning conductors ; a barbed galvanised wire, stretched a foot or two above the conductors and well grounded at distances not exceeding 200 yards, forms a reliable protection.

Generators capable of giving out not less than 1 K.W. (=1·34 H.P.) for each H.P. to be transmitted should be provided ; a low frequency, preferably not exceeding 30 per second, is best for the transmission of power, whilst 60 per second may be looked upon as the highest number admissible ; on the other hand, electric lighting is unsatisfactory unless the frequency exceeds 50 per second, so that it is rarely possible in long distance transmission to use the same current both for power and for lighting unless a rotary transformer is employed. The voltage should be 2,000 to 3,000 for small powers, and 3,000 to 10,000 for large, but the modern tendency seems to be to even exceed the latter figure. There is also a growing tendency to generate the current at the highest possible voltage, and only to use transformers when the quantity

of power to be transmitted or the distance of transmission is so great as to necessitate a tension of over 5,000 volts.

A dynamo of less than 25 H.P. can be wound for a maximum of 1,000 volts						
,,	,,	25 to 50 H.P.	,,	,,	,,	2,000 ,,
,,	,,	50 to 100 H.P.	,,	,,	,,	3,000 ,,
,,	,,	over 100 H.P.	,,	,,	,,	5,000 ,,

The question at what tension a line should be worked and whether it is or is not advantageous to use transformers to attain a high voltage depends on purely economic considerations; broadly, it may be said that the longer the distance the greater the economy that can be realised by the use of transformers and high voltages. The economy of copper in the line that can be thus obtained is shown by the following table of the number of ampères required to generate 1 B.H.P. at the motor pulley at different voltages, allowing a motor efficiency of 90 per cent. and a power factor of 0·75:—

Tension in volts.	Ampères per B.H.P.
1,000	1·11
2,000	0·55
3,000	0·37
5,000	0·222
10,000	0·111

Of the various types of motors there are two systems that are employed in the transmission of power, namely, synchronous and asynchronous. The former require a direct current exciter and commutator, will only start with relatively light loads, have a speed nearly constant under varying loads, and their power factor at full load is nearly equal to unity.

Asynchronous motors need no exciters, will start under full load with two phase and three-phase currents, but with single phase require to be started by means of a special arrangement of condenser, whilst their speed

decreases as the load increases. Their power factor at full load is between 0·65 and 0·85 according to size.

Each type has accordingly its inherent defects and advantages, but upon the whole the preference seems mostly to have lain hitherto with synchronous motors, the higher power factor being apparently considered by most authorities to compensate for the additional trouble in starting them and getting them into step.

Numerous examples of electrical installations for the purpose of driving stamp mills might be quoted, chiefly from the Western States of America, the Transvaal, and New Zealand, as well as other localities.

The Sheba mill of 60 stamps has been driven by electricity for some three years, and the same power has recently been applied to a second mill of the same size, auxiliary steam power being also provided. The 60 stamp mill was estimated to require 162 H.P. to drive it. The power station is five miles from the mill, power being generated by two Victor turbines capable of producing together nearly 400 H.P. The generators are two-phase alternators with exciters capable of producing 150 E.H.P. at a pressure of 3,300 volts. The armatures are stationary 54 inches in internal diameter by 12 inches wide. The field magnets with 16 poles run at 400 revolutions per minute, giving a frequency of 53 per second. The conductors consist of three concentric cables laid in a trench nowhere less than 3 feet deep. Each cable contains two conductors each of 19 wires, 0·057 inch in diameter; the resistance is 8·8 ohms for each five mile length of double cable. Four 50 kilo-watt transformers, reducing the tension down to 100 volts, are employed. The motors to drive the mill are 50 H.P. motors of the induction type,

working with two-phase currents at 100 volts potential; they run at 640 revolutions per minute. The crushers are driven by two 15 H.P. motors, and the vanners by one of the same power. The efficiency of the mill motors is 92 per cent at full load. The cost of the entire plant with additions that enabled it to transmit 350 H.P. seems to have been about £35,000, and in 1897 the cost of electric power and transmission per ton crushed was as follows:—

	s.	d.
Wages, salaries, stores and supplies . .	1	10·170
Repairs and maintenance	—	11·087
Total	2	9·257

This cost is said to have been reduced since then.

At Pilgrim's Rest the Transvaal Gold Estates have completed a large electric installation; here Girard turbines drive three-phase dynamos and generate the current, which is transmitted at a potential of 3,000 volts; this is converted by means of step-down transformers into low tension (120 and 220 volt) currents, by means of which, among other motors, a 65 H.P. three-phase motor driving a 20 stamp mill and a similar 100 H.P. motor driving a 60 stamp mill are supplied with power. The latter motor was found capable of working only 50 stamps, but in all other respects the installation is quite successful.

At Bodie, California, the Standard Mining Company is working both mill and mine by electricity transmitted for a distance of 12½ miles. The generating plant consists of a Pelton wheel coupled direct to the armature shaft of a 120 K.W. alternate current single phase generator running at 860 revolutions per minute; the potential is about 3,400 volts. The conductor consists of a double

line of wire 0·29 inch in diameter, carried on glass insulators, the poles being set 100 feet apart. The mill, which contains 20 stamps with pans, vanners, &c., is driven by a 120 H.P. synchronous motor, a small 10 H.P. Tesla starting motor being used to start the larger machine by means of a friction pulley. But little difficulty is experienced in starting the motor and getting it into step. The motor also runs at 860 revolutions per minute, the speed being reduced by means of a single countershaft to 80 per minute. The efficiency of the generator is 95·5 per cent., and of the motor 93·9 per cent., whilst the line loss is between 8 and 9 per cent. The total efficiency of the system from water power to motor pulley is therefore about 79 per cent. The entire cost of the plant was $38,000 (say £7,700).

In 1896 it was found in California that the average cost of installations transmitting over 1,000 H.P. to distances of between 13 and 25 miles, ranges between $100 and $140 (£20 and £30) per H.P., the charges for maintenance and working (by water power) varying from $13·50 to $30 (£2 16*s*. to £6) per H.P. per annum.

Mills are electrically driven at several mines in New Zealand ; it is said that the first battery in the world to be worked by electric transmission was at Skippers' Creek. All the New Zealand installations are, however, on a small scale. In the Transvaal there are electric transmission plants, in addition to those already referred to, at Moodie's, also driven by water power, and a large plant is now being contemplated on the Witwatersrand to transmit 10,000 H.P., the generating station being steam driven in the last named.

Steam Power.—Even where good and cheap fuel is available, this is far the dearest method of driving a mill,

as will have already appeared from the statistics given. A large number of mills are worked with wood as fuel, and when this can be obtained of good quality at prices ranging from 10*s*. to £1 per cord, as it can in many mining districts, no better fuel can be desired. In tropical countries, however, where a large proportion of the rapidly-grown wood is soft and worthless for fuel, the price is frequently much higher and is often the most expensive item in gold milling. Thus at the El Callao Mine in Venezuela, the cost of fire wood was £2 per cord. In the Johannesburg district coal is available at a cost of about 18*s*. to 20*s*. per ton, but its quality is rather poor, some of it having a steam raising power of less than one-half of that of good Welsh coal. In the case of small mills, that cannot afford a first-class engineer, it is probably advisable to lay down a good simple high-pressure engine, of ample power to do all the work of the mill, and with a boiler somewhat larger than is usually considered sufficient, so as to ensure an ample supply of steam; *e.g.*, for a 10 N.H.P. engine it would be advisable to lay down a 12 N.H.P. boiler, more especially when inferior fuel has to be employed. Large and well-equipped mills should however be driven by engines of the most approved construction of the compound high-pressure type, with a good economical cut-off; condensing engines may in some cases be used with advantage. An ample battery of boilers should be provided, of such capacity that there may always be one standing idle, in order to enable all to be kept well cleaned and in good repair. As a rough general rule it is usual to provide an engine of such size as to give one nominal H.P. for every stamp; *e.g.*, a 20 N.H.P. engine for a 20 stamp mill, &c. If the engine and boiler are good ones they will readily give an I.H.P. three

times as great as their N.H.P., and this rule accordingly gives ample power for driving the stamp-mill, and all other machinery that may be required. In large mills it is well to make the engine house an entirely separate building from the stamp mill, so as to lessen the chance of grease or oil finding its way from the engine to the mill. For large mills, whatever kind of motor be employed, cotton ropes are to be preferred to belting for driving the first motion shaft.

Illumination—It is impossible to expect good work to be done in a mill unless it be thoroughly well lit, so that the mill-men can see what they are doing. A badly lit mill, moreover, gives opportunities for the theft of amalgam that would not occur in a well lit one. There should be plenty of windows in the ends as well as in the roof of the mill, and lighting on the night shift should receive careful consideration. There is no better method of lighting a mill at night than by means of electricity, preferably by powerful (50 to 100 c.p.) incandescent lamps. Not only does electricity give a powerful light, but it has also the immense advantage of not requiring any oil or grease about the mill. As rock-breakers are, generally speaking, only run during the day shift, the power thus set free at night-time can be used to advantage in running a dynamo for electric lighting, for which purpose additional power would not therefore need to be provided.

When oil lamps have to be used they should be large and powerful; it should be one man's business to take them down in the morning, clean, trim, and fill them, and put them up in their places again before the night shift comes on, and a special place should be set apart for this work; in large mills, where an oiler is kept, he

should also attend to the lamps. By this means the risk of oil finding its way into the mortar boxes or on to the plates is decidedly lessened, and it is always an advantage to allow only one man in the mill to handle all the oil and grease, and not to give this man any other work to do that will take him near the copper plates or the mercury.

Mill Sheets.—It is all important for the successful running of a mill that proper complete statistics be kept. For this purpose a slate should be kept hanging in the battery-house upon which the superintendent or head amalgamator can write his instructions, and the mill man in charge of the shift can note down all necessary observations that will enable him properly to fill up his mill-sheet. This he should do every shift, or else in some mills it is the duty of a clerk to come round and copy out the figures on the slate for the past twenty-four hours, and then to put them into a mill book. The mill-sheet should be ruled for a month at time, and have headings somewhat as follows:—

MILL-SHEET FOR THE MONTH ENDING .

Date.	Shift.	Number of stamps at work.	Average speed per minute.	Average length of drop.	Cars of quartz received.	Weight of Amalgam taken from			Stoppages.	Remarks.
						Outside plates.	Inside plates.	Other sources.		

Columns may also be added for the number of concentrators at work and the quantity of concentrates produced daily.

The following is the form of mill sheet recommended by Mr. Nicol Brown :—

WORK DONE FOR THE MONTH OF .

Number of men employed.	Date..	Number of stamps running.	Time.		Drop of stamps.		Cause of stoppage.	Name of ore milled.	Weight of Amalgam.	Concentrates.	Remarks.
			Hours working.	Hours stopped.	Height.	No. per min.					

He adds that the screen used and the height of discharge should be noted once on each sheet, and any change in the same should be recorded under "Remarks."

A useful check upon the mill men consists of a revolution counter attached to the cam shaft. This enables the superintendent to ascertain at once what amount of work the mill has been doing during each shift and to check the speed of the mill and length of stoppages as reported by the mill man. In addition to the mill-sheets, a separate mercury book should always be kept as already recommended, and the results of each clean-up, the total amalgam produced each month, and the weights of sponge and bullion obtained entered in another book kept specially for that purpose. It is only by the careful collection and comparison of accurately recorded statistics concerning all the details of the working that the defects of any system can be ascertained and improvements introduced.

CHAPTER XV

SAMPLING AND ASSAYING OF ORE, TAILINGS, CONCENTRATES, AND BULLION

It is often part of a mill man's duty to assay the ores which he is called upon to treat, and the bullion, concentrates, and tailings which he produces, so as to check the results obtained in the mill. In large mills there is sufficient work to keep an assayer continuously engaged, but in smaller ones it is part of the chief amalgamator's duties, and in any case this is work that he ought to be able to perform if necessary. An assay office is by no means a costly adjunct to a mill, and it is a most important one, as without it scientific work is quite impossible, and in its absence serious losses of gold from preventable causes may go on unchecked because undiscovered.

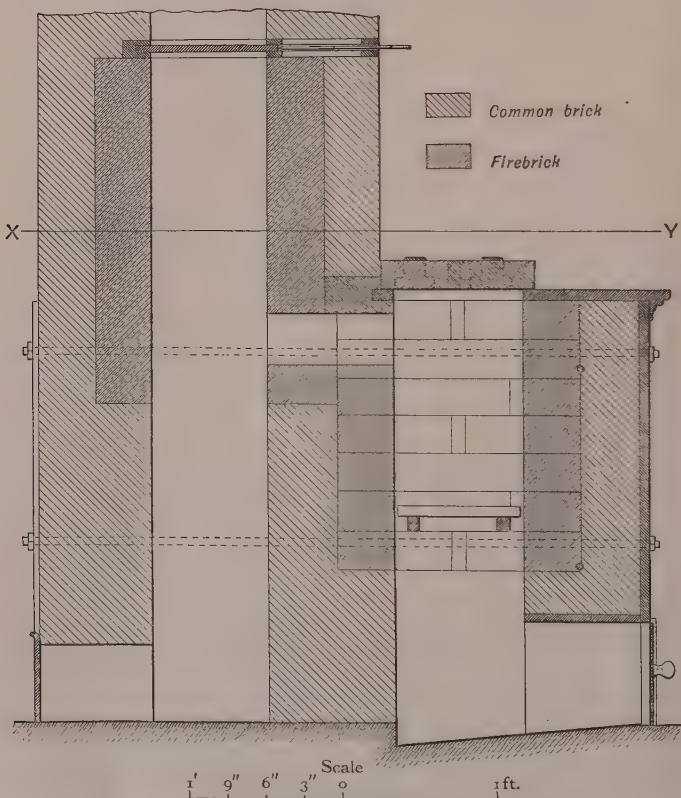
Assay Office.—This should be near the mill for the sake of convenience, being only so far off as not to be affected at all by the vibration of the stamps. It should consist of three rooms: an outer room or shed for sampling and rough work; a furnace room with work bench and ample space for the storage of crucibles, chemicals, &c., and an inner balance room. The furnace room should be fitted with a sink and water supply, and

a well-ventilated draught chamber for the carrying off of objectionable fumes.

The Sampling Shed should have a well-laid floor of either concrete or stout plank, and in the middle there should be a stout iron plate about 4 feet square and $\frac{1}{2}$ inch thick. There should be a large mortar about 2 feet high and 8 inches internal diameter, with a pestle about 4 feet long, and weighing some 50 lbs.; this is advantageously suspended from a strong spiral or other spring attached to one of the rafters. This large mortar may be supported on trunnions on which it can revolve, being kept in place by a couple of folding wedges. There should also be a couple of smaller iron mortars and a couple of Wedgwood ones, one large and one small. In large mills where big samples of ore have frequently to be examined, it is advisable to have a sample grinder driven off some portion of the mill-shafting in a corner of the mill building, as much time and labour can be saved by it. Various forms of sample grinders are made by makers of mining machinery.

Furnaces.—Two kinds of furnace are needed in gold assaying, the ordinary wind furnace for fusions, and the muffle furnace. The best form of wind furnace is shown in Figs. 122 and 123. It is practically a short shaft about ten inches square, lined with fire brick and covered and held together with iron plates. At a depth of about 16 inches below the top a couple of bearer bars are built into the furnace, upon which the fire-bars are supported. These may either be of cast-iron or else of 1 inch square iron bars $9\frac{3}{4}$ inches long, laid about $\frac{1}{2}$ inch apart. Below the fire-bars is the ash-pit which is closed by a light iron door having a sliding shutter by which the admission of air to the ash-pit may be controlled. The furnace flue

should be about 3 inches by 4 in area, with a sliding cast-iron damper. The flue opens into a short stack about

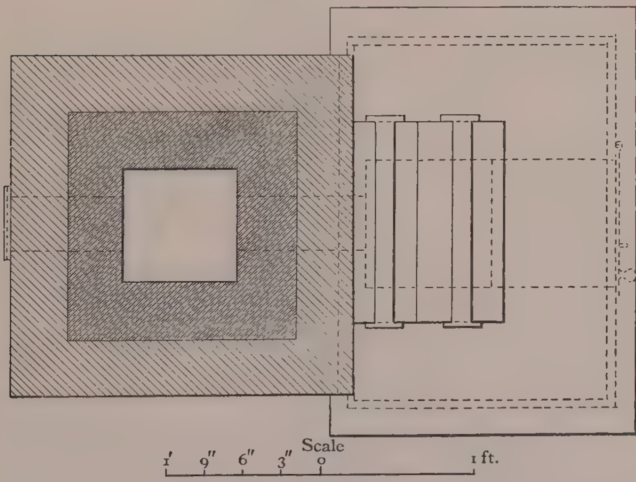


Vertical Section

Fig. 122.

20 feet in height, with which the other furnaces also communicate. The cover consists of two fire-brick tiles

strapped with iron. Instead of the above square furnace which has to be built on the spot, the Black furnace may be employed which can be bought ready for setting to work. The Black furnace is well adapted to the work of the gold-assayer, except that the cover usually furnished with it is quite unsuitable. A piece of boiler-iron about 2 feet square should have a circular hole cut in it equal



Section Plan on X.Y.
FIG. 123.

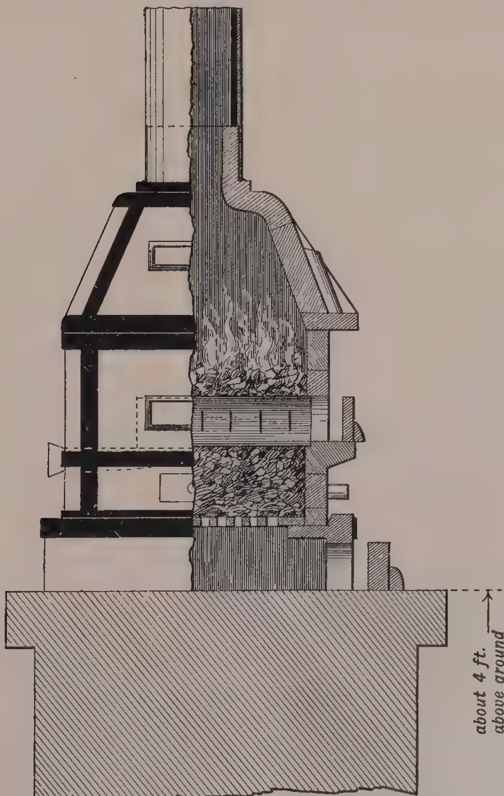
to the size of the furnace top, and by means of three iron brackets this should be riveted to the furnace so as to form a firm level top; upon this can rest a cover like that used for the square furnace. The thin sheet-iron stove-pipe usually supplied with the Black furnace is far too light and burns out rapidly; the first length of 5 or 6 feet next to the furnace at any rate should be made of

stout iron $\frac{1}{8}$ inch thick, and the elbow may with advantage be even thicker. A circular damper working on a pivot should be inserted in the pipe for controlling the draught.

The best form of muffle furnace is that shown in Fig. 124. It is manufactured by the Morgan Crucible Company, Limited, of Battersea. The most generally useful size is that taking a muffle size D ($8\frac{1}{2}$ inches $\times$ 5 inches $\times$ $3\frac{1}{4}$ inches). This furnace should be set upon a brick pier of such a height that the muffle itself is a little lower than the height of the assayer's shoulder. It should be furnished with a pipe of stout sheet-iron which connects it with the stack, or if preferred it may have its separate stack in the shape of an iron stove-pipe about 10 feet in height. If preferred, a muffle furnace may be built on the spot. It is shaped like a shallow square melting furnace which has been raised some 3 feet from the ground, and which has an opening in front some 2 inches above the fire-bars, which corresponds to the mouth of the muffle.

Balance.—This should be a first-class instrument, such as is constructed expressly for gold-assaying. It need not be a large one: one capable of carrying 30 grains in each pan and, when thus loaded, of turning distinctly with 0·0005 grain is sufficient for all purposes. A portable assay balance, manufactured by Oertling, may also be used; it presents the advantage of unusual compactness. A set of gold assay weights from 20 grains down to 0·01 grain, and two riders for sliding along the graduated beam weighing respectively 0·1 and 0·05 grain are required. In gold assaying it is advisable to use no other standard of weight than the troy grain and the troy ounce. As all bullion has to be weighed by the

latter unit, it is best not to attempt to use any other in the assay office. Not only are calculations facilitated by



Scale

1 ft. 9" 6" 3" 0 1 ft.

FIG. 124.

this means, but confusion and possible error are avoided. The advantages claimed for the assay ton system of

weighing are in reality almost imaginary, seeing that by the system here recommended, a simple reference to the tables given further on, and which should be hung up in every assay office, avoid the necessity for any calculation.

No description of the assay balance is here given, as the assayer will learn more about its construction in five minutes' intelligent examination of it than he could from many pages of letterpress. Moreover it may be taken for granted that none will commence assaying without the benefit of a little practical advice from an experienced assayer. The following hints as to the management of the balance may however be useful.

The balance case should always be kept closed when not in use. A dish containing a little quicklime should be kept inside the balance case, and renewed from time to time. A camel-hair brush should also be kept in the balance case for dusting the beam and pans when necessary, the brush being used for no other purpose than this. The substances to be weighed are always put in the same pan (usually the left). In weighing a button of metal this should be put into the left pan, and a weight supposed to be greater than the weight of the button placed in the opposite one; should it prove to be too small, a larger one must be tried until a weight is found greater than that of the button. This is then replaced by the next smaller weight, and so on until it is found between which two consecutive numbers (being integers of grains) the real weight of the button lies. The smaller number being then left in the pan, the tenths of a grain weights are then tried similarly in descending order, and the first decimal of the required weight is thus determined, the remaining two figures

being obtained by sliding the 0·1 grain rider along the graduated beam, when its position, as soon as the balance is in equilibrium, will give the number of hundredths and thousandths of a grain that go to make up the total weight of the button. When very minute buttons are being weighed, as often occurs in tailings assays, it is advisable to use a rider weighing only 0·05 grain and to divide the result thus indicated by 2 for the true weight of the button. In bullion assays, it is advisable to weigh by substitution, placing a constant weight larger than that of the body to be weighed in one pan, and the body itself in the other, weights being added to the latter until the balance is in equilibrium; the difference between the weights in the respective pans is then the weight of the body. When the balance is nearly in equilibrium, the oscillations of the pointer on either side of the zero mark of the scale must be read; when these are equal, the balance is in equilibrium. It is never safe to assume that the balance is in equilibrium because the beam when released does not incline to either side; it must be caused to vibrate, and the amplitude of its vibrations read on the scale. In using the balance care must be taken that the beam is released and arrested gently, without any shock or jar that might damage this delicate instrument. The weights must never be handled with the fingers, but always with the forceps specially provided for this purpose, and which must be used for no other.

For weighing out ores, &c., a pair of scales is required, one of the pans of which should be movable. The pans should not be less than $4\frac{1}{2}$ inches in diameter, and the scales should indicate distinctly $\frac{1}{10}$ grain when loaded with 4,000 grains. It is useful to keep a strong set of

common scales or a spring-balance weighing, say, to 56 lbs., in the sampling room, and as a general rule the bullion balance is kept in the balance room of the assay office, the assay furnace being used for melting down the gold sponge.

Sampling.—This is one of the most important operations in making assays, and one that is too frequently neglected. An assayer may have to sample :—

(*a*) The ore as it comes to the mill.

(*b*) Ore already extracted, and lying in heaps or bins.

(*c*) Ore standing in the mine.

(*a*) When the ore is received at the mill, a shovelful should be taken by the rock-breaker man from the cars as they come from the mine, with a frequency depending on the size of the cars and the size of the sample required. Thus a sample may be taken from every other car when daily assays of the ore are made, or from every tenth car when only a weekly assay is desired. It is, perhaps, better to have a shovelful taken out of the ore feeder hopper at stated intervals, as by this means a fairer sample of large and small ore is obtained. In either case all the ore so taken out is thrown into a bin or piled in a heap in one corner of the mill floor.

(*b*) It is scarcely possible to get a really fair average sample from a large heap unless the whole heap is turned over. This can but rarely be done, so that the best plan consists in cutting a trench about 2 feet wide right across the heap from side to side, and throwing all the ore that comes out of this trench into a heap. If an average sample of several piles of ore has to be taken, each one must be treated in this way, and all the ore so obtained thrown together.

(*c*) It is sometimes necessary to take a sample from the face of a level or stope in the mine. When this has to be done, the floor should be first of all cleaned up as thoroughly as possible; ore is then broken down by means of the pick and wedges from the whole surface of the face, taking care that it is taken uniformly from every part. All the sample thus obtained is trammed out and piled in a heap.

The result of any of these three operations will be a pile of ore that should fairly represent the average composition of the entire mass that has been sampled; it should in no case be less than half a ton, and may amount to several tons. It is first broken down to pass a 2-inch ring if the sample exceeds a ton, or a 1-inch ring if under a ton. It is then thoroughly mixed by means of a long-handled shovel and made into a conical heap; the top is then flattened down until a circular heap is produced, having the shape of a short truncated cone. This heap is then divided into four equal parts; this may be done either by marking lines on its surface by means of a pick, or by laying on it a cross-shaped piece of wood, each arm of the cross being of the same length as the radius of the bottom of the heap. Two of the four quarters thus formed are then thrown aside, either of the diagonally opposite pairs being selected. Thus, in Fig. 125, either *a* and *d* or *b* and *c* may be thrown aside. In the half thus left, all large pieces of ore are again broken down, the heap is thoroughly mixed again, and the above operation of halving is repeated until the pile is reduced to 2 or 3 cwt. The rest of the ore is then returned to the ore bins, and the selected heap is sent into the sampling office. An old die does very well for breaking the ore on, a 4 lb.

breaking-hammer being used. The above operation may be performed by a mechanical sampler which breaks and subdivides a parcel of ore into any given number of equal parts, or the breaking may be done by means of a small rock-breaker, the sampling being still performed by hand. The sample as sent to the sampling-house is then broken down in the large mortar, and the same operation of quartering down continued until a sample weighing about 7 lbs. is produced. The sample, which should by this time be crushed fine enough to pass a sieve of 8 holes to the linear inch, is still further reduced till a fair sample weighing about half a pound is obtained for fire assay.

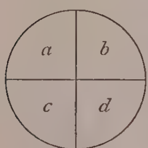


FIG. 125.

The remainder of the 7 lb. sample is kept for panning and amalgamation assays, if these are needed. They are not as a rule required when ore, the character of which is well known, has to be assayed, but all new samples should be subjected to them, before the fire assay is proceeded with, as very much valuable information regarding its character is thus obtained.

Sampling Tailings.—It is of the utmost importance that constant and careful assays be made of the tailings as they run to waste. For this purpose a convenient spot should be arranged in the waste tailings launder, where a drop of about a foot can be given so as to admit of placing a bucket, or better still, a small square box which exactly fits the launder beneath the issuing stream of tailings at stated intervals. This vessel should be filled but not allowed to overflow, as in this case more or less concentration of the heavier part of the tailings would ensue. Such a sample should be

taken at intervals of say three hours, and all the pulp thus obtained poured into a tank or tub. The product of each day's work is allowed to settle completely, and when the supernatant water is perfectly clear, it is syphoned off. The settled pulp and slimes are then scraped or rinsed out through a plug-hole in the bottom of the containing vessel into an iron pan, in which the remaining water is evaporated off so as to get a perfectly dry sample. This is then thoroughly mixed and taken to the sampling room, where it is divided down just as already explained in the case of ore, until about $\frac{1}{2}$ lb. is left, which is then ready for fire assay. If preferred, the tailings obtained throughout the week may be mixed and a fair sample taken of the parcel so obtained. Assays of tailings should always be made at least once a week, and in large mills every day. In some mills a simple automatic tailings sampler is used, which consists of a short length of launder pivoted under the main launder; in its normal position this short movable launder discharges into another long one, which carries the tailings away to waste. The movable spout is so connected either with a clockwork arrangement, or else with a miniature water wheel driven by the current of the tailings themselves, as to be deflected at given intervals for a short period, during which it discharges the current of pulp into a suitably placed vessel. Another ingenious tailings sampler is on the principle of the Barker's mill with a number of spouts all of different lengths, only one of which can deliver into the vessel in which the sample is being collected. The further treatment of the sample of pulp so obtained is the same as when the sample is taken by hand. In some mills samples are also taken of the pulp as it leaves the

battery screens by passing a small bucket along the front of the lip from end to end and thus catching a portion of the issuing pulp; here again care must be taken to avoid any concentration of the coarser or heavier portions. These samples give but little information when inside amalgamation is employed, and are hence not often taken. A better practice is to sample the pulp as it runs to the concentrators, as well as the tailings after they have left them, so as to control the efficiency of these machines. It may here be stated that assays of tailings should be supplemented by their occasional examination under the lower powers of a microscope, in order to see, if possible, in what forms, the gold is being lost. Sometimes it becomes necessary to sample heaps of tailings; this may be done in the same way as described above for heaps of ore, namely, by cutting a trench through the heap and sampling down the material so obtained. Heaps of tailings or tailings settled in tanks or pits may also be sampled by driving iron tubes with a lower cutting edge through the mass from top to bottom at uniform distances apart along lines right across the mass; the cores of tailings left in the tubes form the sample. Piles of concentrates are sampled in the same way, but as it is a difficult matter to obtain good representative average samples of these, and as their value is usually high, it is better to take two quite independent samples from each heap, to sample these down and assay them quite independently of each other and then take the mean of the two results.

Ore Assays.

Amalgamation Assay.—A convenient quantity for operating on is from two to three pounds. A good plan is to

take a weight equal exactly to 2·24 lbs. (avoirdupois), this being exactly one-thousandth part of a ton, unless results are to be returned by the short ton of 2,000 lbs., when 2 lbs. will have to be taken. This weight of 2·24 lbs. is obtained by placing $2\frac{1}{4}$ lbs. (avoirdupois) in one scale pan, and 70 grains (troy) in the other. Shot is then poured into the second pan until it exactly balances, when the weight of the shot will be the required weight; or if preferred, a leaden or brass weight may be thus prepared and kept for use. This quantity of ore is then weighed out. It is crushed in one of the smaller iron mortars and sieved through a sieve of 60 holes to the linear inch. The best sieve for this purpose is a cylindrical box sieve, about 5 inches in diameter and of the same depth, fitted with a proper cover to prevent loss of ore. If the ore contains coarse gold, this will not pass through the sieve, but will be left on it, as it will flatten under the pestle instead of being crushed by it. There may also be left on the sieve a few pieces of iron derived from the pestle or mortar, which may be removed by means of the magnet. It is advisable to heat these metallic residues with a little dilute nitric acid in a small porcelain basin, so as to remove any adhering crusts of oxide of iron, &c., which might prevent amalgamation; the residues are then washed two or three times by decantation and thrown into the parcel of crushed ore. This is then transferred to a Wedgwood ware mortar (or, if preferred, an iron one can be used), of about 12 inches in diameter, and mixed with enough warm water to make a stiff paste. About an ounce of dry mercury, thoroughly purified and free from gold, is then carefully weighed out; it is warmed in a small hard-glass flask and a piece of clean sodium about the size of a pea thrown in.

The mercury thus highly charged with sodium is then thrown into the mortar with constant stirring, the flask being, if necessary, rinsed out with water, so that none of the mercury is lost. The entire mass is then ground steadily for an hour, when amalgamation should be quite complete. The paste so formed is then transferred in three or four lots to a prospecting-pan, and carefully panned up into a tub in which the tailings are caught. The mercury is collected and transferred to a small dish; if it is much floured and refuses to run readily into globules it may be stirred with a small piece of sodium held in the end of a glass tube, which will cause it to run together. The mercury is then washed with a light stream of clear water until it is quite clean, the water is poured off and the mercury dried by means of blotting paper. It is then reweighed, when its loss of weight should not exceed 5 per cent. If it is greater than this, the tailings must be panned repeatedly until the excess of loss is made good. If this cannot be done, the assay must be rejected and a fresh one commenced. The mercury is next transferred to a small gold annealing cup which has been carefully black-leaded inside, covered with a porcelain or fire-clay cover, and the mercury evaporated off at a gentle heat. This may be done by placing it in the mouth of a muffle, when the fumes will be drawn up the stack, or else in a light sheet iron retort made expressly for this purpose. When all the mecury is completely volatilised, about 50 grains of assay lead are thrown into the crucible; this is then pushed a little way into the muffle so as to melt the lead; the crucible is then withdrawn, its contents mixed by giving it a gentle rotary motion, and poured into a small mould such as is used in fire assays (see page 537). The

resulting button of lead is then cupelled, as will be subsequently explained, and the button of bullion so obtained is weighed. It may be assumed, without sensible error, that the mercury lost in the operation carried with it the same proportion of gold as is contained in the mercury which is recovered. Hence the gold contents of the ore will be found by multiplying the weight of bullion obtained by the weight of mercury originally taken and dividing the product by the difference between the weight of mercury recovered and the weight of bullion found. This figure multiplied by 1,000 gives the richness of the ore in free gold in grains per ton. Thus, suppose, for example, that a lot of 2·24 lbs. of ore was thus treated with 450 grains of mercury; after amalgamation 442 grains of auriferous mercury were collected which yielded 10·02 grains of gold bullion. Then the amount of amalgamable free gold present in the ore per ton of 2,240 lbs. would be:

$$\frac{450}{442-10} \times 10{\cdot}02 \times 1{,}000 = \frac{10{,}020 \times 450}{432} = 10{,}437 \text{ grs.} = \overset{\text{oz.}}{21}\ \overset{\text{dwt.}}{14}\ \overset{\text{grs.}}{21} \text{ per ton.}$$

Panning Assay.—The prospecting-pan is too well-known an implement to need much description; it consists of a pan with sloping sides and slightly convex bottom, one or two grooves being run round the upper edge of the sides, which are turned over so as to stiffen and strengthen the pan and to enable it to be handled conveniently. The tinned iron pan of the Australian miner is of no use to the mill man. These pans should be made of thin sheet steel or of first-class Russia sheet iron, having a highly planished surface. They should be perfectly smooth inside, and kept free from rust. Pans are often used for panning up sands containing amalgam or mercury, and it will be found very difficult to free

these from all traces of mercury except by heat. It is, therefore, advisable to keep a special set of pans for panning all material containing mercury, and to distinguish them clearly from the others, which must never be used for this purpose. A good plan is to use pans with two grooves for the former, and pans with one groove round their upper edge for panning material free from mercury. Pans of two or three different sizes should be kept at hand. A most useful adjunct to the pans is the batea (not to be confounded with the mechanical batea, page 437) which is a shallow wooden dish having the shape of a small segment of a sphere of large diameter. It is made of hard wood cut across the grain, and is especially useful in washing ores containing fine flour gold or very finely divided sulphurets. The fine heavy material clings to the comparatively rough surface of the wood, and is more easily caught in the batea than in the metal pan. The batea should never be used for material carrying mercury or amalgam. For such the horn is sometimes used; this consists of a deep oval spoon-shaped bowl cut out of a large ox-horn, in which concentration is easily and rapidly performed. For vanning out sulphurets from crushed ore, the Cornish vanning-shovel may be used, or else the white enamelled iron plaque manufactured for this purpose by Messrs. Fraser and Chalmers, Limited. The mode of using any of these instruments is very difficult of description though comparatively easy of execution, and involves a knack that can only be acquired by practice. It is of the utmost importance to the mill man to be thoroughly expert in their use, and he should lose no opportunity of working with them all until he has acquired the art. For a panning assay the same weight

of ore should be taken as for the amalgamation assay. This is then pounded in the mortar and sieved through a sieve of 40 holes to the linear inch, any gold in the residue left on the sieve being set aside. The pounded ore is then panned, the tailings being allowed to flow into a larger pan and panned again so that nothing may be lost. This operation will show at once whether the ore is rich in sulphurets, and what their nature is. The visible gold should be panned as free as possible from all adhering sulphurets, taking care, however, that none of it is lost. The pan is then dried and its contents, together with any coarse pieces left on the sieve, brushed into a piece of lead foil; this is rolled up, melted with a little assay lead in a scorifier, and then cupelled; these processes will be presently described in detail. They result in the production of a button or prill of gold, which can be weighed, and thus the free gold present in the ore determined. The tailings produced in the panning operation should then be panned over two or three times to collect all the sulphurets present in the ore. These should be dried and weighed, and their percentage in the ore determined; if weighed in grains troy, this calculation is easily made by remembering that the pound avoirdupois contains 7,000 grains, so that the amount originally taken was 15,680 or 14,000 grains according as the long ton or the short ton is the standard of calculation. These sulphurets may then be assayed by fire assay to determine their value. If preferred, the tailings from the amalgamation assay may be panned up so as to collect the sulphurets present in them, which may then be dried, weighed, and assayed. As, however, some of these are apt to be slimed in the process of amalgamation, the results are less reliable

than when a separate panning test is made. Another method consists in not separating the free gold from the sulphurets, but in treating both together by fire assay, and determining the total gold present in them.

Fire Assay.

Theory of Fire Assays.—Let it be supposed that we have to operate on an ore consisting only of pure silica and free gold, both in a fine state of division. If this ore be mixed intimately with carbonate of soda, litharge, and charcoal in proper proportions, and the mixture heated in a crucible to the fusion point, the following reactions would take place:—

1. The charcoal would reduce a portion of the litharge to metallic lead—it being taken for granted that not enough charcoal is added to reduce the whole of the litharge.

2. Part of the silica would combine with the unreduced litharge to form silicate of lead.

3. The excess of silica would combine with the soda of the carbonate of soda to form silicate of soda, carbonic acid being evolved, and any excess of carbonate of soda remaining unchanged.

4. The lead produced would alloy with all the metallic gold present. The contents of the crucible would therefore ultimately form two layers; a lower one of auriferous lead, and an upper one of slag, consisting of silicates of soda and lead, together with, perhaps, some carbonate of soda. In practice this latter would attack the crucible, forming also silicate of soda. As a general rule, 100 grains of litharge require for their reduction about 3·5 grains of ordinary charcoal powder. Silicates of varying

compositions are produced when different proportions of silica and carbonate of soda are melted together, but it may be taken that to form a readily fusible silicate, 100 grains of silica require about 150 grains of carbonate of soda.

Gold ores, however, almost always contain other ingredients besides gold and silica, the most important of these being oxides of iron and various sulphurets. The oxides of iron are mostly the peroxide Fe_2O_3, but sometimes magnetic oxide Fe_3O_4; these both, when fused with silica, form silicates of the protoxide, so that in the case of the former oxide, one equivalent of oxygen (16 parts) is liberated by every 160 parts, and in the latter by every 232 parts of oxide. This oxygen will oxidise a certain proportion of the carbon present in the crucible charge, so that for every 100 grains of peroxide of iron present there will have to be added about 7 grains more of charcoal, and for every 100 grains of magnetic oxide about 5 grains more. When sulphurets are present, both the sulphur and the metal exercise a reducing action, the former being oxidised to sulphurous acid, and the latter to a protoxide, which combines with silica to form a silicate. In the case of iron pyrites, which is the most plentifully occurring sulphide, 100 grains of it would reduce no less than 930 grains of litharge, thus taking the place of about 32·5 grains of charcoal. When there is much pyrites present in the ore it is advisable to substitute red lead for litharge, the former having a high oxidising power; 100 grains of pyrites reducing about 710 grains of red lead. There are, however, various other methods by which this difficulty can be overcome.

Preparation of the Ore.—The $\frac{1}{2}$ pound of ore which

has been the ultimate result of sampling down, as previously described, is ground in an iron mortar until the whole of it has passed through a sieve of 100 holes to the linear inch. Great care must be taken that the whole of the sample is put through without any loss. Any metallic particles that may be present will, of course, be beaten flat, and will not pass through the sieve. These are shaken out on to a sheet of glazed paper, and carefully put on one side for the present, forming the so-called "metallics." The sifted ore is then weighed, and its weight recorded in the assay book. The sifted ore is then thoroughly mixed on a sheet of glazed paper or, better, of stout American cloth by means of a spatula, and transferred to a wide-mouth stoppered bottle.

Fusion.—The following mixture is then weighed out :—

Ore	500	grains.
Litharge	500	,,
Carbonate of soda	400	,,
Charcoal	15	,,
Dried borax	200	,,

The litharge should be first weighed out accurately and placed on a sheet of American cloth, then the ore, weighed as exactly as the scales will admit of, is placed on top of the litharge, upon this the charcoal, finally the carbonate of soda and half the borax. The last two need not be weighed accurately, but can be measured out in spoons known to hold about the required amount. The charge is then mixed most intimately by means of a spatula, until the colour is quite uniform throughout, and no separate particles of the ingredients can be distinguished by the eye. The mixture is then transferred to a crucible about 5 inches in height, and 3 in diameter (sizes *F* and *G* of the Morgan Crucible Company,

Limited, are the most suitable for this work). The remaining 100 grains of borax are then poured on to the sheet, rubbed over it with the spatula so as to take up any particles of the charge that may have been left behind and poured on to the mixture in the crucible so as to form a cover for it. The charge should then not more than two-thirds fill the crucible. Some assayers prefer to heat the crucible first, in which case the charge is poured into it by means of an assayer's scoop, the 100 grains of borax being placed behind the mixture so as to sweep any particles of the former that may have been left behind into the crucible.

Meanwhile the fire should have received proper attention and have burnt well through. The crucible should be placed in the fire with the tongs shown in Fig. 126, which are the best patterns for handling the crucible with. Tongs shaped as in Fig. 127 are useful for packing in fuel, lifting the crucible cover and light work generally. The pots should be put well down in the fire, covered with a proper cover and then packed with fuel. Coke broken into pieces about 2-inches cube is the best fuel, but charcoal can be used in case of need; the latter fuel should be screened through a $\frac{3}{4}$-inch screen so as to remove all small stuff and to ensure a good draught. The heat is raised slowly, the damper being about $\frac{1}{3}$ open for the first ten minutes, by which time the charge should commence to frit; the heat may then be raised gradually for five minutes and urged for five minutes more, by which time the crucible should be at a full red heat, and its contents thoroughly liquefied and in a state of tranquil fusion. The crucible is then lifted out and the fused mass poured slowly and steadily into one or other of the moulds shown in Fig. 128, the

mould having been previously well blackleaded inside and warmed. These moulds are made of cast-iron, and the cavities turned out carefully and very smoothly polished. It is as well to have several of them at hand. When thoroughly set, the contents of the mould are turned out on to an iron plate and allowed to cool. A plate of cast-iron about 1 foot square, and $1\frac{1}{4}$ inches thick, with a smooth surface, should be set on the work-bench near the furnaces for this purpose. The button thus turned out consists of a lower layer of lead and an upper layer of slag; the latter should be glassy,

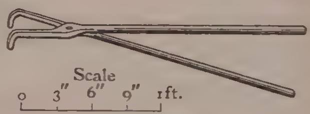


FIG. 126.

greenish, brownish, or black, quite uniform in texture and colour, free from spots and streakiness, and contain no included shots of lead. It is broken off from the lead by blows from a light fitter's hammer, weighing $\frac{1}{2}$ to $\frac{3}{4}$ lb. When the bulk of the slag is removed, the lead button is held edgeways on the plate and hammered all round the edge; this causes its upper and lower faces to

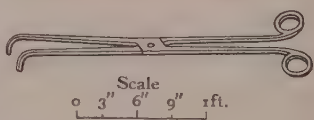


FIG. 127.

wrinkle up, and loosens the remaining fragments of the slag. The lead button is then well brushed with a stiff brush, such as a small nail-brush, until quite clean and free from every particle of slag. The slag should be broken up and examined for shots of lead; if any are found they must be added to the main button. The lead button should be marked with its number, or in some other distinctive manner, by means of a punch, such mark being entered in the margin of the assayer's

note-book. The lead, which should be perfectly malleable, is then hammered into a roughly cubical shape and weighed. Its weight should be between 350 and 400 grains. If less than the former amount the assay must be repeated, with a proportionately larger quantity of charcoal. If the ore was found on panning to contain a large amount of oxide of iron, which is usually sufficiently indicated by its colour, the above proportion of charcoal in the mixture must be increased in the ratio already pointed out; in this case, also, 100 or even 200

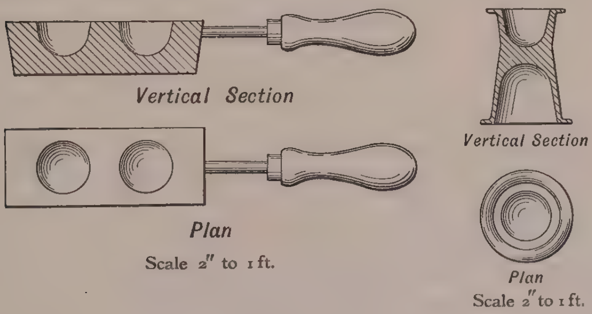


FIG. 128.

grains of the carbonate of soda may be advantageously replaced by borax, the latter having the property of readily fluxing oxide of iron by forming a fusible borate, whilst some of the oxide of iron, acting as a base and combining with the silica present, takes the place of carbonate of soda.

If there is a large proportion of sulphurets present, or if concentrates are being assayed, the best and simplest plan is to destroy these sulphides by calcination. For this purpose a clay roasting dish, which is a shallow saucer-shaped clay vessel, about 4 inches in diameter, or

as large as will conveniently go into the muffle, is rubbed over inside with plumbago or red ochre. The charge of ore, carefully weighed out, is transferred to this dish, and this is then placed in the muffle, which should be barely red-hot. The sulphur in the charge almost immediately takes fire, burning at the surface of the ore with its characteristic bluish flame when present in large proportion. The ore must be gently stirred with an iron rod, about $\frac{1}{8}$ inch in diameter, the end of which is beaten into a thin flat spatula-shaped point, $\frac{3}{8}$ inch broad and 2 inches long. Care must be taken to raise the heat very gradually, and to keep continuously stirring, so as to keep the charge of ore from clotting. When no more sulphur fumes can be seen, and when the ore leaves off glowing brightly at each stroke of the stirring rod, the heat can be raised to redness; stirring must, however, be continued until no more smell of sulphurous acid can be perceived on taking the dish out of the muffle, and the ore appears quite dull when it is stirred. The calcined ore is then allowed to cool, and when cold is mixed with the same charge as the raw ore, and fused in the same way. It must not, however, be forgotten that the iron pyrites originally present has now become converted into iron peroxide, so that the proportion of reducing agent must be increased, and 200 grains of carbonate of soda and 400 of dried borax substituted for the quantities in the normal charge.

When concentrates are being assayed they must be calcined as above. The calcined substance will then consist principally of oxide of iron and contain very little silica. In order to form a fusible slag, silica must therefore be supplied and this is best done in the form of ordinary green bottle glass ground up and passed through

a sieve of 80 holes to the inch. 500 grains of concentrates are thoroughly calcined and when cold mixed intimately with :—

Litharge	500 grains.	
Charcoal	30	,,
Ground Glass	400	,,
Dried borax	400	,,

100 grains of Borax being as usual reserved for a cover.

In case of concentrates containing much galena it is scarcely possible to calcine with any approach to completeness on account of the low fusibility of lead sulphide. In this case it will be best not to calcine, but to prepare the following mixture with the raw concentrates :—

Ore (or concentrates)	500 grains	
Red lead	400 ,,	(or less, in proportion to the amount of galena present.)
Red argol (crude tartar)	30 ,,	
Carbonate of Soda	400 ,,	
Borax	100 ,,	

the borax being used to cover the mixture, which is transferred to the crucible in the usual way. A couple of pieces of clean hoop iron 4 inches long $\frac{3}{4}$ inch broad and $\frac{1}{8}$ inch thick are then stuck into the mixture, the pot is put low down and well covered, care being taken that the fire is not too hot. At the end of ten minutes all the mixture should be melted down; the mass is then well stirred with the pieces of hoop iron and the pot covered again and left for another ten minutes. The pieces of iron are then taken out, the fire is urged for a few minutes more and the contents of the crucible are then poured in the usual way. The pieces of hoop iron should be examined for adherent shots of lead, and if any are found they must be added to the main button.

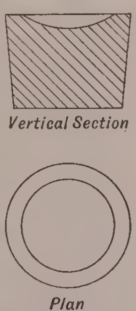
When charcoal is used as a fuel it is always advisable

to charge the mixture into a pot previously heated, and the charge should be such as to be very readily fusible. For this purpose crude tartar can be used to replace charcoal, 4 to 5 grains of the former being generally about equal to 1 of the latter. The proportion of borax may also be increased by another 100 grains.

In every case the ultimate result of the fusion is the production of a button of lead with which all the gold originally present in the ore is now alloyed.

Cupellation.—When a piece of lead alloyed with gold or silver, or both, is heated in contact with air, it melts, and the surface of the molten lead becomes covered with a layer of oxide. If this oxide be removed as fast as formed, the process of oxidation will continue until all the lead is removed and the precious metals alone are left. This is the principle of the process of cupellation, the removal of the lead oxide being effected by the device of conducting the operation in a porous vessel, not attacked by molten lead oxide, and which can therefore absorb the oxide as soon as formed, just as blotting paper absorbs water. If copper is present in the alloy in not too great quantity it is oxidised, and its oxide is apparently dissolved in the molten lead oxide and carried into the porous vessel with it. The vessel in which the operation is conducted is called a cupel; it is made of bone ash and generally has the shape shown in Fig. 129; its exact shape is not, however, very material, and several different patterns are used. Cupels are made from bone ash, carefully calcined at a comparatively low temperature until it becomes quite white. It is rarely necessary and never advisable for an assayer to prepare his own. In fact cupels for gold assay are best purchased. French (Deleuil) cupels are generally quite satisfactory. If it is

necessary to make them, this is done by means of a cupel mould, which consists of a cylinder of iron or gun-metal pierced by a cylindrical hole equal to the diameter of the cupel it is proposed to make. Into it fits a piston-like piece, the lower end of which is turned into shape so as to form the cavity of the cupel; great care must be taken to have this as smoothly finished as possible. Coarsely ground bone ash, which will pass through a sieve of about 20 holes to the linear inch is used. This is stirred up in water, allowed to settle for a few minutes, and the milky water carrying any very finely divided bone ash in suspension is poured off, and allowed to stand, so as to deposit this suspended bone ash. Each grade is dried separately. The coarse bone ash is then mixed with just sufficient water to make it cohere when squeezed in the hand. The mould is filled with this, topped with a little of the dry, fine, levigated bone ash, and the piston driven down by a few blows of a mallet. As the success of the cupels depends largely upon the due regulation of the pressure at which they are made, it is better to use a proper press instead of a mallet for their manufacture. When made, the cupels are pushed out of the mould from below, and set aside to dry.



Scale one half.
FIG. 129.

When the lead buttons produced by the fusion process are ready for cupellation, the requisite number of cupels are placed in the muffle, the fire having previously been lit. The size of the cupel to be used is determined by the weight of the button of lead; the cupel should be rather heavier than the lead button it is to carry. The best fuel for the muffle furnace is coke broken into pieces

of 1 to $1\frac{1}{2}$ inches cube, but in case of necessity charcoal can be used, although the operation is rendered rather more troublesome. The charcoal should be in pieces between $\frac{1}{2}$ and $1\frac{1}{2}$ inch cube, and free from dust or fine stuff. The muffle should be brought to a full red heat, the cupels examined to see that they contain no particles of dirt, &c., and then the buttons of lead are rapidly charged by means of a pair of long light tongs, each into its respective cupel, the position of which in the muffle should be noted down; the muffle door is then closed. In a few minutes all the buttons should have melted and the door should be very slightly opened; it is best to have a muffle door made in two pieces, a lower part and an upper one, the latter being much the smaller. If the upper part of the muffle door is now drawn back about $\frac{1}{4}$ of an inch, a sufficient current of air will be admitted into the muffle. The heat must be kept up to a bright redness, fuel being added if required, but before the fire burns down as far as the top of the muffle, as this would be chilled by contact with the cold fuel. It is best to avoid making up a fire during the course of a cupellation, but this is not always possible when charcoal is the fuel used. The temperature of the muffle is right when the molten lead looks bright and luminous and markedly convex, when the spots of oxide are seen to move rapidly over the face to the edges, and when fumes rise from the cupels without hanging over them and disappear before they reach the top of the muffle. As the operation proceeds the beads rapidly diminish in size and become globular in shape; presently they show a series of iridescent rings, and then suddenly brighten, the rapid motion of the surface appearing to cease abruptly. The

cupels should be left in a few minutes more and then withdrawn; no harm is done by leaving them in the muffle for some time in case some of the cupellations are finished before the others. The cupels are withdrawn by means of light specially-made cupel tongs and allowed to cool. The little buttons or "prills" of gold and silver should be globular in shape, slightly crystalline on the upper surface, frosted-looking underneath, and should not adhere to the cupel. They are lifted up by means of a light pair of long-nosed pliers with half-round points, squeezed and flattened on an anvil. This should be a small stake anvil with a face about 2 inches square, set in a solid block of wood; it may with advantage be surrounded with a sheet-iron tray about 1 foot square with a rim 1 inch deep; the middle of this tray has a square hole through which the stake of the anvil just passes, by means of which the tray is kept in position. The hammer used should have slightly-rounded faces and weigh about $\frac{1}{4}$ lb.; both anvil and hammer faces must be kept thoroughly clean and bright. The lower surfaces of the prills are well brushed with a small stiff tooth-brush to remove any adhering particles of bone ash, and they are then carefully weighed in the assay balance and their weights noted with the most scrupulous accuracy.

Treatment of "Metallics."—Meantime the "metallics" obtained by sifting and crushing the ore have also to be treated. Any particles of iron are first removed by the magnet, and the residue, which should now consist only of native gold, is wrapped in about 50 grains of lead-foil and cupelled. The prill thus obtained is also carefully weighed.

Parting the Prills.—The prills still contain silver in

addition to gold, and this has to be removed by the process of parting. When an alloy of gold and silver is boiled in nitric acid, the silver can be completely dissolved out, provided that the alloy contains at least $2\frac{1}{2}$ times as much silver as gold. When working upon an unknown ore, the proportion of silver present must be guessed by the colour of the prill, and enough added to bring it up to three times the supposed weight of gold. When the colour of the alloy is white a weight of silver equal to that of the prill should be added. It is important that the silver used for this purpose be quite pure. Pure assay silver is prepared as follows: Coin silver is flattened under the hammer, cut up by means of a pair of stout shears, and dissolved in a mixture of one part of nitric acid to two parts of water. The solution is evaporated to dryness, the dry mass heated to fusion until it blackens, and the saline residue dissolved in warm distilled water and filtered. The filter is well washed, and the filtrate diluted to two pints for each ounce of silver present. The solution is then heated to boiling, and hydrochloric acid added with constant stirring until the precipitation is complete. The mixture is set aside in a warm place and allowed to settle. When the solution is perfectly clear it is tested with a drop of hydrochloric acid to make sure that all the silver has been precipitated. When this has been accomplished, the clear solution is carefully syphoned off, the vessel is again filled with boiling water, stirred, allowed to settle, and the supernatant fluid again syphoned off when clear, the process being continued until the wash-water is free from all traces of acid. The chloride of silver is then washed into an evaporating basin, drained as thoroughly as possible and dried. The

dry chloride is mixed with an equal bulk of carbonate of soda, and fused in a clay crucible such as is used for gold assays, but which the mixture must not more than two-thirds fill. When it is in a state of quiet fusion it is poured into any convenient ingot mould. The ingot of silver is cleaned from adhering slag, washed, scoured with a little fine sand, and is then rolled into a thin strip with constant annealing if there is a flatting mill at hand; if not, it must be hammered out as thin as possible, well annealed, and cut into small pieces with the hand shears.

Each of the prills resulting from cupellation, together with the proper weight of assay silver, is wrapped in 50 grains of lead-foil, which is rolled into a little ball and cupelled as before. The resulting buttons are carefully cleaned and flattened out on the anvil. Each prill is then dropped into about $\frac{1}{4}$ oz. of No. 1 parting acid (which has been previously raised to boiling in a small test tube) and kept boiling for ten minutes. This No. 1 parting acid consists of 1 part of nitric acid to $2\frac{1}{2}$ parts of distilled water. It is advisable to add a drop of silver nitrate solution to every bottle of parting acid, to make sure that it shall be free from any trace of chlorine. After boiling for ten minutes, the acid is poured off and the residue boiled for ten minutes more with No. 2 parting acid, consisting of equal parts of nitric acid and water. This acid is then poured off, and the pure gold which remains in the bottom of the test tube is washed two or three times with distilled water. The test tube is then filled with water, covered with a small gold annealing cup, which is a crucible made of porous clay, and the cup and test tube inverted together. The gold

then falls into the cup, the test tube is removed, the water in the cup poured off, and the gold thoroughly dried and annealed by heating to redness. It is then weighed with all due precautions and its weight recorded.

A point that must not be left out of consideration when reporting the results is that all lead products, such as litharge and red lead, contain some silver; the amount of this has to be subtracted from the total weight of the prill obtained. To determine this amount, 500 grains of red lead or litharge are heated in a small crucible with about 20 grains of charcoal, for ten minutes, when the action should be complete. The fused mass is then poured, the button of lead cleaned from any adherent slag, and cupelled in the usual way. As a general rule, 500 grains of ordinary litharge or red lead will contain from 0·01 to 0·02 grain of silver.

Calculation of the Results.—This is an extremely simple matter by the help of Table No. II. From the weight of the prill of bullion the weight of silver present in the oxide of lead is subtracted; the difference is the amount of bullion (gold and silver) actually yielded by 500 grains of ore. Reference to the table will show what this is equivalent to in ounces, pennyweights, and grains per ton. Similarly the quantity of pure gold per ton is taken out from the table, the difference between the two being the yield in silver per ton. It is always advisable to make gold assays in duplicate, taking the mean of the two, which should not differ greatly from each other, as the true result. The following example will perhaps render clear the mode of returning the assay results:—

	I.		II.
Weight of prill of bullion	0·046 gr.	.	0·047 gr.
Weight of silver in 500 grains of litharge .	0·013 ,,	.	0·013 ,,
Weight of bullion produced by the ore . .	0·033 ,,	.	0·034 ,,
Average bullion contents	0·0335 grain =		
2 oz. 3 dwt. 18·6 grns. per ton of 2,240 lbs.[1]			
Weight of pure gold obtained	0·027 gr.	.	0·028 gr.

		oz.	dwt.	grs.
Average (fine gold contents)	0·0275 =	1	15	22·4
Silver in the ore	0·0060 =	0	7	20·2

Of course, to the amount of gold and silver thus obtained the amount in the "metallics" must be added. A simple calculation will show how much of this "metallic" gold and silver would be yielded by 500 grains, since the amount yielded by the original assay sample (the weight of which has been recorded in grains) is known. The corresponding proportion of ounces, &c., per ton is taken out from the table and added to that previously obtained.

If, for instance, the assay above quoted had been made on a portion of a parcel weighing 3,760 grains, the metallics from which had yielded on cupellation a bullion prill of 0·018 grains and after parting a button of fine gold weighing 0·014 grains we should have

3,760 : 0·018 : : 500 : 0·0024
3,760 : 0·014 : : 500 : 0·0019.

And from the tables we find that these figures so obtained correspond to

	oz.	dwt	grns.
Bullion per ton	0	3	3·3
Fine gold per ton	0	2	11·6
Silver per ton	0	0	15·7

[1] This is taken out as follows from Table II. :—

	oz.	dwt.	grns.
0·030	1	19	4·8
0·003	0	3	22·1
0·0005	0	0	15·7
0·0335	2	3	18·6

which amounts must be added to the respective ones previously obtained.

On the Continent of Europe it is usual to report such assays by the number of grammes of gold to the metric ton; Tables IV. and V. are conversion tables to enable returns thus reported to be converted into ounces, &c., per long or short ton and vice versa.

Scorification.—When very rich products, such as mattes or high grade concentrates, have to be assayed, the method of scorification is sometimes resorted to. This consists in exposing the substance to be assayed mixed with granulated lead to a high temperature in the muffle, the mixture being contained in a small saucer-shaped vessel of good fire clay, known as a scorifier. The following mixture is made :—

Substance to be assayed	50 grains.
Granulated assay lead	250 ,,

This is well mixed and transferred to the scorifier and covered with 250 grains more of granulated lead, on top of which are placed 50 grains of borax. The muffle must be heated to a white heat, and the charged scorifiers are then placed in it by means of specially shaped tongs. The muffle door is closed until the charge is completely melted down, which should take 10 to 15 minutes. The door is then slightly drawn back so as to admit a current of air, and the operation continued as long as the molten lead in the scorifier continues to oxidise. In this operation, volatile substances, such as arsenic and sulphur, are almost completely expelled; oxidisable metals, such as iron, copper, and bismuth, are oxidised and carried into the slag, together with a large amount of oxide of lead, any siliceous matter present forming silicates. When the bath of molten lead is

completely covered by a layer of slag, about 3 grains of powdered anthracite wrapped in a little piece of tissue paper are dropped into each scorifier, with the object of cleaning the slag. After another five minutes the scorifier is withdrawn by means of the scorifier tongs and the contents poured into a button mould. The lead is cleaned from slag in the usual way; it should weigh from 250 to 300 grains and be quite soft and malleable. If it is not soft or if it is above 300 grains in weight it must be put back into the muffle in a fresh scorifier (the same one may be used if not too much corroded) with 20 grains of borax and the process repeated until the desired result is attained. The button of lead is then cupelled in the usual way. The weight of the prill obtained is of course multiplied by 10 in calculating the quantities per ton. Scorification should be resorted to whenever the button of lead produced in the crucible assay is too heavy or is not perfectly soft. It may also be used with advantage for treating the concentrates produced in the panning assay if their weight does not exceed 100 grains.

Tailings Assay.—Tailings are assayed in exactly the same way as ores. In this case, however, as the gold contents are usually very small, it is better to work on a double quantity of ore and fluxes either in a larger crucible or else in two separate lots; the resulting button or buttons of lead are scorified down until a button weighing about 350 grains is obtained which can then be cupelled as usual. Naturally in calculating the quantities per ton the results must be divided by 2.

Tabulating Results.—It is evident that a complete series of assays of all the products of the mill should show exactly what has become of all the gold contents

of the ore. In making these calculations it is, however, necessary to take care that the errors due to moisture in the ore are eliminated, otherwise the results will not agree. For this it is necessary that the fire assays should be conducted upon dry ore; the sample prepared for fire assay is dried in an iron pan over a fire, at a low temperature, care being taken that no oxidation is set up, and the same thing must be done with the concentrates. It has already been pointed out that if the weight of ore going into the mill is determined by measurement, as recommended, and if the weight of dry ore corresponding to the standard of measurement is determined, the mill returns will then practically be made on the dry ore sent to the mill.

The fire assay gives the total contents of gold per ton of dry ore. The results of milling give the weight of gold actually obtained per ton of dry ore. The percentage yield of concentrates per ton of dry ore is similarly known, and their assay value is also known. The tailings assays, being made on dry tailings, are similarly reducible to the same standard, and these three figures added together should be equal to the assay value of the ore. From the assay value of the concentrates it is known what loss is incurred in treating them. When the tailings are further treated, as for instance by the cyanide process, it must first be found by experiment on a large scale what weight of tailings ready for treatment is yielded by a ton of ore, and then the losses in the retreatment of the tailings can be accurately determined. Of course assays of the exhausted tailings should also be made continuously, so as to check the efficacy of the process adopted for their treatment. Careful tabulation of all these results will not only show at what stages of the process gold is being lost, but forms

a good check on and consequent preventive against stealage in the mill.

Cyanidation Assay.—It is sometimes necessary to determine what percentage of the gold can be extracted by cyanidation from a given ore or tailings. The material should be ground to pass through a sieve of 100 holes to the linear inch; 1,000 grains are weighed out into a 16 ounce wide-mouthed bottle fitted with a well-ground stopper; 27 grains of coarsely crushed potassic cyanide are thrown in and then 6 ounces of water carefully measured. Common water is perhaps better than distilled water for this purpose. The above proportion—4·5 grains of commercial cyanide to the ounce of water—gives almost exactly a 1 per cent. solution of KCy. The bottle is well agitated, and allowed to stand in a dark place, not too cold, for 72 hours; it should be shaken up from time to time and the stopper taken out so as to allow access of air to the solution. At the end of the above time, the clear solution is decanted off through a coarse filter into an evaporating basin about 7 inches in diameter, and evaporated, best on a water bath. The ore remaining is washed three times with about 3 ounces of water each time, and the washings gradually transferred to the same basin. A filter pump will be found of great service in this part of the work, as the proper washing of a clayey ore is a slow process without it.

The solution is evaporated to complete dryness, the residue loosened from the dish by means of a steel spatula and pulverised in the dish with a porcelain pestle. It is then mixed in the dish with:—

Litharge	500 grains.
Argol	50 ,,

This is transferred to a suitable crucible and the dish

rinsed out twice with 100 grains of borax each time, rubbed down in it with the pestle. This borax is then charged into the crucible as a cover to the previous mixture. It is heated in a rather slow fire, poured and the resulting lead button cleaned and cupelled as usual.

Another method consists in concentrating the solution as above to two or three ounces, and then transferring this to a little narrow basin or boat made by closing up the ends of a strip of lead-foil; this boat is best placed on a similar strip of lead-foil and covered with a third piece to prevent spirting; the three pieces of lead-foil should together weigh between 300 and 400 grains. The solution is evaporated to complete dryness in the leaden boat, and when dry all the lead is thrown into a scorifier and melted up; the lead button obtained is cleaned and cupelled as usual.

In either case the prill produced is weighed and parted as above. It must not be forgotten that in taking out the results per ton, the figures must be divided by two. The object of the above assay is merely to find out whether the ore in question is amenable to cyanide treatment. A complete cyanidation test is much more elaborate. The degree of fineness of the ore, strength of solution and period of treatment, that give the best results technically and economically have to be determined by means of percolation tests on rather larger samples, whilst the acidity and quantity of "cyanide" substances present must also be determined. This is, however, work that had better be left to the trained chemist; the simple cyanidation assay here described is generally sufficient for the purposes of the assayer.

Bullion Assays.—It is finally necessary to determine the value of the bullion produced. For this purpose each

bar of bullion should be assayed. A sample is taken by chipping off small pieces from diagonally opposite corners of each ingot, or else from drillings. In the latter case a fine twist drill of about $\frac{3}{16}$ inch diameter should be used. A hole should be drilled in the middle of the bottom of the ingot, and two holes, one at each end of the upper face, and the drillings thus obtained be mixed together. When pieces are chipped off, they should be flattened in the flatting mills, or beaten out as thin as possible with a hammer, and cut into small pieces by means of shears. Annealing should be avoided, because the composition of the alloy, if rich in copper, is apt to be slightly affected by it. For the same reason it is not advisable to melt together the chips or borings taken into one piece. The only correct way of sampling low grade bullion is by a sample dipped out when the alloy is molten; a good plan is to thrust the stem of a clay pipe to the bottom of the crucible, plug the pipe with a little clay, and take the wire of gold thus got as a sample. If the approximate composition of the bullion is known the assay proper may be at once proceeded with, if not, a preliminary assay is made on 5 grains. This amount is carefully weighed out; $2\frac{1}{2}$ times its weight of assay silver is also accurately weighed out, and the two metals are wrapped in about 50 grains of lead-foil and cupelled. When the cupellation is finished the cupel is slowly withdrawn from the muffle, and the button cleaned and weighed. The difference between its weight and that of the bullion and silver originally taken is set down as copper. The button is then rolled or hammered flat, and boiled in two lots of nitric acid, as is done with the prills obtained in the ore assay. The residual fine gold is washed, dried, heated to redness, and

weighed. Its weight is set down as pure gold; the difference between the weight originally taken and the sum of the weights of copper and gold is put down as silver. It is necessary to have in readiness a quantity of pure gold, which is best prepared as follows:—A quantity of the gold cornets from previous assays is dissolved in nitro-hydrochloric acid at a gentle heat, and evaporated over the water bath till the solution commences to crystallise. It is then diluted with warm distilled water till the solution contains not more than ½ ounce of gold to the pint. This is allowed to stand for some days in a large beaker closely covered up. The clear fluid is then carefully decanted from any residue that may have been deposited into a large dish or beaker. It is heated to boiling, and mixed with a hot strong solution of oxalic acid, an amount of oxalic acid being used equal to that of the gold originally taken. After boiling for a few minutes, the solution is set aside until all the gold has settled and the supernatant fluid, which must react strongly acid, is colourless. The completeness of the precipitation may be ascertained by taking out a small quantity of the solution and testing it with a few drops of chloride of tin. When it is complete, the clear fluid is syphoned off and the precipitated gold washed by decantation some half a dozen times with hot distilled water. It is then thrown on to a large filter, washed thoroughly with hot water, and dried. The dried gold is transferred to a suitable clay crucible, the filter burnt, and the ashes added, the whole covered with a little dried borax and melted. The resulting button of fine gold is cleaned from adherent slag, well scrubbed, and rolled out in the flatting mill into thin foil with constant annealing.

Two lots of bullion each of 10 grains are carefully

weighed out; the amount of pure gold supposed to be present in each is calculated, and so much pure silver is added that its weight *plus* that present in the bullion shall be equal to $2\frac{1}{2}$ times the weight of the gold present. The metals are then wrapped in a piece of lead-foil, weighing from 50 to 100 grains. A piece of pure gold equal in weight to the weight of gold supposed to be present in the 10-grain lot of bullion is then weighed out, together with as much assay silver as will equal the total weight of that present in the bullion *plus* that added, and as much pure electrotype copper as will equal the weight of that supposed to be present in the 10 grains of bullion. These metals are then wrapped up in the same weight of lead-foil as was used for the assay. This latter constitutes a so-called "check" assay, and is made up so as to represent as exactly as possible the composition of the bullion to be assayed. The two bullion assays and the check assay are then cupelled simultaneously under as exactly as possible the same conditions. When the cupellation is finished, each cupel is covered by inverting an old cupel over it, and the covered cupels are then drawn towards the door of the muffle, which is left open so that they may cool very gradually. When cold, the three buttons are detached from the cupels, carefully cleaned, and accurately weighed. They are then flattened on a clean bright anvil by means of a hammer about 12 lbs. in weight with rounded faces, which must be kept very smooth and bright. They are then annealed and rolled in the flatting mill into strips about 2 to $2\frac{1}{2}$ inches long, and $\frac{3}{8}$ inch wide. If there are no flatting mills at hand, the buttons may, with a little practice, be drawn out by the hammer alone, but the flatting mills are preferable. The strips should be

annealed once or twice during the operation, care being taken that their edges do not crack or become rough. When drawn out they are again annealed and rolled up between the finger and thumb into spiral coils about $\frac{1}{4}$ inch in diameter; it is best to keep the side of the button which was next to the cupel, and which presents a somewhat duller appearance than the other, for the outer side of the spiral.

The dissolving out of the silver is done in assay flasks; these are small hard glass flasks of a capacity of about 5 ounces. Into each are poured about 2 ounces of No. 1 parting acid, which is then heated. As soon as it has reached the boiling point, one of the spiral coils, or "cornets" as they are called, is dropped into each flask, together with a fragment of burnt fire-clay about the size of a pea to prevent bumping. After 15 minutes' boiling, the acid is poured off, the cornets are washed, and boiled for 15 minutes more with 2 ounces of No. 2 parting acid. This is then poured off and the cornets again washed with distilled water. The acid and washings should be poured into a vessel reserved for the purpose and the silver recovered from time to time by precipitating with hydrochloric acid and reducing the chloride of silver so produced. A gold annealing cup is next inverted over each flask, which should have been filled to the top with water, and when the latter is inverted with the cup, the gold cornet falls gently through the column of water into the cup. The flask is then removed, as much water as possible poured off from the cornet, and the cups with their contents dried. They are then heated to low redness in the muffle, and when cold are carefully weighed. If the weight of the check piece has undergone no change, the mean of the weights of the two assay pieces repre-

sents the quantity of pure old present in 10 grains of bullion. If the check piece has gained in weight, its increase must be subtracted from the mean of the weights of the assay pieces; and if it has lost, the decrease must be added to get the correct quantity of gold present. Similarly the mean of the loss in weight of the two assay pieces after cupellation represents the amount of copper or other impurities present in 10 grains of the bullion, which must be similarly corrected by the loss of weight—here there cannot be a gain—of the check piece.

The silver is determined by difference. The results are reported millesimally by multiplying by 100. An example will make this plainer:—

Preliminary Assay—

Bullion taken	5·01 grains	
Fine silver	12·60 ,,	
	17·61 ,,	
Weight after cupellation	17·44 ,,	
Loss	0·17 grain	= 0·339 grains of copper in 10 grains of bullion.
Weight after parting . .	4·450 ,,	= 8·882 ,, gold in 10 grains of bullion.
Difference		= 0·779 grains of silver in 10 grains of bullion.
		10·000 ,,

Actual Assay.—If the 10 grains of bullion to be taken for the assay proper contain 8·882 grains of gold it will be necessary to alloy this with $\frac{5}{2} \times 8{\cdot}82 = 22{\cdot}205$ grains of silver. But the bullion already contains 0·779 grains, so there need only be added $22{\cdot}205 - 0{\cdot}179 = 21{\cdot}426$ grains.

ASSAYS.

	No. I.	No. II.
Bullion taken	100·00 grains . .	10·000 grains.
Assay silver	21·427 ,, . .	21·428 ,,
Total weight . .	31·427 ,, . .	31·428 ,,

Check piece.

Fine gold	8·882 grains.
Assay silver	22·205 ,,
Pure copper	0·339 ,,
Total weight . .	31·426 ,,

	Assay piece No. I.	Assay piece No. II.	Check piece.
Weight after cupellation	31·087 grains .	31·086 grains .	31·078 grains.
Loss of weight . . .	0·340 ,, .	0·342 ,, .	0·348 ,,
Average loss of assay pieces			0·341 grains.
Excess loss of check piece 0·348 – 0·339 . .			0·009 ,,
Actual amount of copper			0·332 ,,
Parts of copper per mil.			33·2 ,,

	Assay piece No. I.	Assay piece No. II.	Check piece.
Weight after parting . .	8·874 grains .	8·875 grains .	8·883 grains.
Average weight of gold in assay pieces	8·8745 grains		
Excess of weight (surcharge) of check piece, 8·883 – 8·882 . . .	0·0010 ,,		
Actual amount of gold .	8·8735 ,,		
Parts of gold per mil. . .	887·35 ,,		

Silver by difference . . . 1000 – (887·35 + 33·20) = 1000 – 920·55 = 79·45.

Composition of the bullion per mil.

Gold	887·35
Silver	79·45
Copper	33·20
	1000·00

When a bar has been assayed it should be carefully weighed. Its weight in ounces and decimals of an

ounce troy should be stamped upon it, together with its fineness in parts of gold per 1,000 and the initials of the assayer, a proper set of steel punches being kept in the assay office for this purpose. When its fineness is known, the value per ounce troy can be found from Table No. III., which gives both the full Mint value and the Bank value. For instance, the Mint value of the gold per ounce troy of the bullion assayed in the above example is taken out as follows from the table:—

	£	s.	d.
800	3	7	11·56
80	0	6	9·56
7	0	0	7·14
0·3	0	0	0·31
0·05	0	0	0·05
887·35	3	15	4·62

The absolute value of gold is fixed in this country by the Coinage Act, 1870 (33 Vict. Chap. 10), which provides that the standard weight of the sovereign shall be 123·27447 Imperial grains, and its standard fineness 916·6 per mil. The minting value of gold is thus fixed, whilst the Act further enacts that the Mint shall coin free of charge any bullion brought to it, provided that the bullion does not need refining in order to bring it to standard fineness, the Mint being thus bound to coin but not to refine for the public. The Bank Charter Act, 1844 (7 and 8 Vict. Chap. 32) compels the Bank of England to buy any bullion that may be presented to it for sale at the rate of £3 17s. 9*d*. per ounce of standard gold, provided, however, that the bullion must be assayed at the expense of the person tendering it. The minimum Bank rate for gold is thus fixed, the difference between Bank and Mint rate (about 1½*d*. per ounce of

standard gold) being for the Bank's commission, and for recouping loss of interest, &c. According to the needs of the market the Bank may pay at times a higher price for gold bullion than the standard rate. The usual charges upon bullion are :—

For melting	. .	$\frac{1}{4}d.$ per ounce of bullion.
,, refining	. .	$4d.$,, ,,
,, assaying	. .	$4s.\ 0d.$ per bar.

The value of the bar is calculated on the assay, subject, however, to certain deductions if the bullion is very base, as is generally the case with cyanide gold precipitated by zinc. These deductions are as follows :—

From assay of base bullion over 800 fine, deduct 2 parts per mil.
,, ,, ,, between 700 and 800 fine, deduct 3 parts per mil.
,, ,, ,, ,, 600 ,, 700 ,, ,, 4 ,, ,,
,, ,, ,, under 600 fine, deduct 5 parts per mil.

On the other hand an extra premium of a $\frac{1}{2}d.$ per ounce of standard gold is paid by refiners on all gold bullion containing only silver and copper in addition to gold. Cyanide gold being irregular in composition and difficult to assay correctly, duplicate sets of assays are made on it, thus doubling the cost of assaying.

The above is the general practice, the charges of different bullion brokers and refiners varying very little from it. Whilst the absolute Mint and Bank values of gold are thus fixed amounts, the actual amount realised depends not only upon the amount of fine gold present, but also upon the fineness of the bullion and the state of the market. The value so calculated refers to the gold only. To determine the value of the silver present, its quantity must be determined by multiplying the weight of the bar by the parts of silver per mil, and dividing by 1,000; the weight of silver is then rated at the

current market price of fine silver per ounce. Gold bullion shipped to England is generally consigned to the Bank.

The great value of the bullion assay for the mill man lies in the fact that he is thus enabled to reduce all his calculations to one common standard, namely, ounces, &c., of fine gold yielded by the ton of dried ore; the value of careful tabulation of all the data obtained in milling has already been repeatedly insisted on, and when all these are reduced to this uniform standard, all the elements for exact and scientific calculations are attainable, and thus the foundation laid for accurate and economical work in the various operations of gold milling.

APPENDICES AND TABLES

APPENDIX A

ON THE CAM CURVE

The proper construction of the cam curve is a matter of such great importance to the successful working of the stamp-mill that I have found it advisable to fully investigate its geometrical properties, and propose here to give a short account of these, so as to enable any mill man to work out all problems connected with this portion of the subject, and to find out for himself whether his cams are of the best possible shape, and therefore working under proper economical conditions, or whether he is losing power and straining the structure of the mill owing to defects in their design.

For the proof of several of the following propositions I am indebted to the kind assistance of Mr. A. C. Waters, who has also revised this section for me.

It has already been stated (page 202) that the curve always employed for the cam is an involute to the circle, and the method there given for setting out the cam is that for describing an involute, the general definition of which is that it is the locus of the end of a string that is being unwound from a circle, the string being always maintained tangential to the circle, and having, moreover, in certain cases, the power of uniformly lengthening or shortening as it is unwound. The definition which I shall here employ is a different one, but I propose to prove later on that the curve generated is in either case the same :—

Let a given finite straight line revolve with uniform angular velocity round one of its extremities as a centre, carrying at its other extremity a perpendicular, along which a point is travelling in the opposite direction with uniform linear velocity; this point will describe an involute to a circle.

The only elements that are required to fix the shape and dimensions of the curve according to this definition are the length of the given finite straight line and the relation existing between the given angular and linear velocities. The circle, an involute to which is thus generated, is the circle described by the moving extremity of the given straight line, and the perpendicular is, of course, always tangential to it. In practice this perpendicular remains fixed and is vertical, being in fact the line of motion of the point lifted (that is to say, it is the central line of the stamp stems), whilst the curved cam surface revolves with the given angular velocity in the opposite direction to that of the radial straight line of the definition, lifting by its revolution the tappet with the given linear velocity. Moreover, the length of the given finite straight line (the radius of the generating circle) is determined, being equal to the horizontal distance between the axes of the cam shaft and the stamp stems.

It is always possible to find a circle, having the fixed extremity of the given line for a centre, of such radius that the arc on this circle included by the angular distance traversed by a radius thereof in any period of time shall be equal to the linear distance travelled by the point along the line of motion in the same time. This circle is called the "circle of equal velocities," and its radius is found as follows: If h be the distance along the line of motion traversed by the point in the unit of time, a° the angle described during the same time, and r the radius of the circle of equal velocities—

$$\frac{2\pi r}{h} = \frac{360}{a^\circ}$$

$$r = \frac{180h}{\pi a} \qquad (1)$$

Now, if d be the radius of the generating circle, d may be either greater than, equal to, or less than r; accordingly three

types of involute curve exist, corresponding to these three ratios, and known as the curtate, normal, and prolate involutes respectively. It will now be necessary to investigate which of these three forms of curve is the best suited to the needs of the stamp-mill. Taking the above definition, it appears that the cam curve must be an involute of some kind, in which the axis of the stamp stem corresponds to the line of rectilinear motion, whilst the centre of the generating circle is in the axis of the cam shaft. The line of motion being vertical in practice, the cam curve has

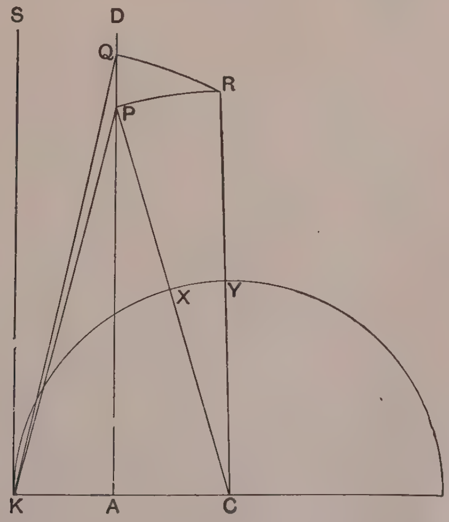


FIG. 130.

to communicate motion to the horizontal tappet by contact therewith ; it is required to determine under which conditions the maximum amount of force can be transferred from the centre of rotation to the line of vertical motion. Evidently a curve, the tangent to which is always horizontal, will be the curve that best fulfils these conditions.

Taking first the general case of prolate and curtate invo-

lutes, Figs. 130 and 131, let C be the centre of revolution, and AD the given perpendicular line of rectilnear motion, CA being thus equal to d. With centre C and radius r draw the circle of equal velocities KXY; at K draw KS, the vertical tangent to the circle, and parallel consequently to AD. Now, let a point on the surface of the tappet move upwards from P in the line AD

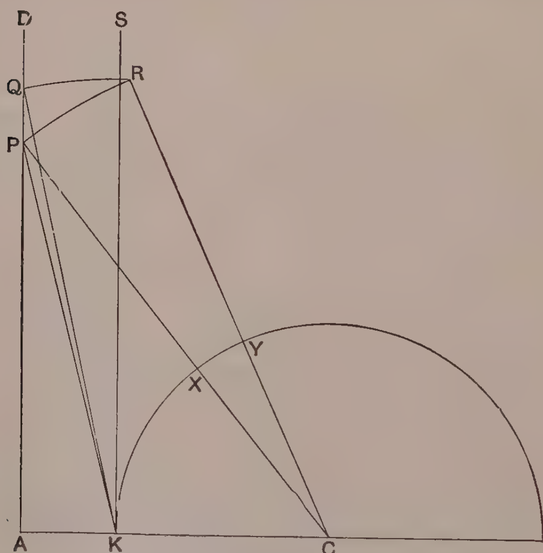


FIG. 131.

through the infinitely small space PQ; draw the angle PCR which shall be equal to the angular motion of the cam corresponding to the rectilinear distance PQ, PR being a portion of a circle described with centre C; then R will have coincided with P at the commencement of the movement of the point to Q, and will have traversed the arc PR, whilst the point has moved up to Q, QR being an element of the involute curve, the revolution of

which from P to R through the angle PCR, has produced the lift PQ. Let CP, CR cut the circle of equal velocities in X and Y respectively ; join KQ, KP.

Since the distance PQ is supposed to be infinitely small, the triangle PQR may be looked upon as ultimately rectilinear, and QR will ultimately become the direction of the tangent to the curve at point Q.

The angle $QPR = 180° - (RPC + APC)$, and since PR is a segment of a circle, RPC will ultimately become a right angle. Hence $QPR = 180° - (90° + APC) = 90° - APC$.

But $KCP = 90° - APB$.

Therefore $QPR = KCP$.

By construction $QP = XY$ and $\dfrac{PR}{PQ} = \dfrac{PR}{XY} = \dfrac{CP}{CX} = \dfrac{CP}{CK}$.

Since in the triangles PQR, CKP, the two sides PQ, PR are proportional to the sides CK, CP, and the included angles QPR, KCP are equal, therefore the triangles are similar, and the angle $PQR = PKC = 90° \mp PKS$, the upper or lower sign being employed according as the prolate or the curtate involute, Fig. 130 or 131, is referred to.

When P and Q are infinitely near together, the angle PKS ultimately $= QKS = AQK$.

Therefore $PQR = 90° \mp AQK$.

But $KQR = PQR \pm AQK = 90° \mp AQK \pm AQK$.

Or $KQR = 90°$.

Which result may be expressed in words by the statement that the tangent to an involute curve at any point on the line of motion is perpendicular to the straight line drawn from that point to a point on the circle of equal velocities, which is on a horizontal radius of that circle, and in the same direction from the centre thereof, as is the line of rectilinear motion. As this result is independent of the ratio of r and d, it is true for all the forms of the involute. If, therefore, either a prolate or a curtate involute be employed for the cam curve, the tangent to this curve would always be inclined to the horizontal face of the tappet, and part of the force that ought to be employed in lifting the weight of the stamp would be wasted by being resolved into a horizontal

thrust either towards or away from the centre of the cam shaft. In the former case the cam would jam against the tappet, whilst in the latter no harm would be done beyond straining the guides and wasting power. it is only in the case of the normal involute that the tangent to the curve, being perpendicular, in that special case, to the line of motion, would always be horizontal, and therefore no lateral thrust at all would be set up. It is accordingly this curve that is always employed for the cam, and it is this curve the setting out of which is illustrated in Fig. 41, page 206. To obviate, however, the slightest risk of producing a curve at all prolate, it is advisable to make it slightly, but only very slightly, curtate. In equation (1)

$$r = \frac{180h}{\pi a}$$

and

$$a^\circ = \frac{h180^\circ}{\pi r} \qquad (2)$$

In the normal involute $r=d$, and this equation may be written $a^\circ = \frac{h180}{\pi d}$. Now let d' be slightly greater than r; then a' will become slightly less than a. Hence to fulfil the above condition of a very slightly curtate involute, it will be well in practice to adopt a value for a *very slightly less* than that given by equation (2) *e.g.* in setting out the cam curve illustrated on page 206, it would be advisable to adopt instead of $91^\circ{\cdot}42'$ (the value there found for a) a smaller value such as $91^\circ{\cdot}35'$, and to take for $\frac{180^\circ}{\pi r}$ (there found to be equal to $13^\circ{\cdot}6'$) the value of $13^\circ{\cdot}5'$, so as to avoid all risk of producing a prolate curve. I have obtained excellent results in practice by the adoption of this principle.

As already stated, it follows from the general proposition respecting the tangent to the involute, that the tangent to the normal involute must always be horizontal, because in the normal involute $d=r$ or $CA=CK$; hence the line KS coincides with the line AD, which becomes in this case a tangent to

the circle of equal velocities at the point on the horizontal radius of this circle.

This proposition can, however, be proved independently, and this separate proof will be found useful because some of the equations obtained in the course thereof will be afterwards applied in calculating the length of the involute curve.

In Fig. 132, with centre C and radius r describe as before the

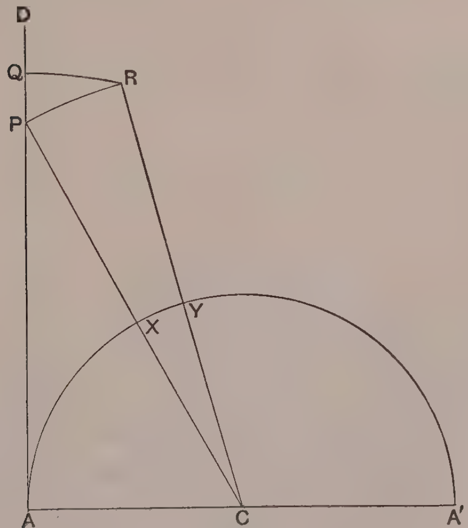


Fig. 132.

circle of equal velocities AXY. Since r is now equal to d, the line of motion AD is now the perpendicular tangent to the circle at A; as before let a point of the tappet be lifted from P through the infinitely small space PQ, let QR be the element of the involute curve corresponding to this lift, which has been produced by the rotation of the cam through the angle PCR, PR being a segment of a circle described with centre C. Let CP CR cut the circle AXY in X and Y respectively.

$$\text{Then } PQ = XY \text{ and } \frac{PR}{QP} = \frac{PR}{XY} = \frac{CP}{CX} = \frac{CP}{AC}$$

and the angle $QPR = 180° - (RPC + APC)$.

But ultimately $RPC = 90°$, and PR, QR become straight lines ; therefore $QPR = 180° - (90° + APC) = 90° - APC = ACP$.

In the triangles QPR, ACP, the two sides QP, PR are proportional to the two sides AC, CP and the included angles QPR, ACP are equal. Hence the triangles are similar and $PQR = CAP = 90°$. That is to say the tangent to the curve at the point Q is perpendicular to the vertical line AD, that is, is horizontal.

Further let $AP = k$.

Then since the triangles QPR, ACP are similar

$$\frac{QR}{PQ} = \frac{AP}{AC} = \frac{k}{r}$$

$$QR = \frac{k}{r} PQ \qquad (3)$$

this equation giving the ratio of PQ to QR for an element of the curve.

In Fig. 133 let a normal involute be employed to lift a horizontal tappet from B to D.

Let the vertical line BD be divided into a very great number of very small equal spaces BQ_1, Q_1Q_2, Q_2Q_3 . . . Q_mD, and draw as before the corresponding arcs BR, Q_1R_1, Q_2R_2 . . . Q_mR_m, which are the arcs described round the common centre C by the respective points of the cam curve which come into contact with the tappet at the points B, Q_1, Q_2, . . . Q_m, whilst Q_1R, Q_2R_1, Q_3R_2 . . . DR_m, are successive elements of the cam curve, each corresponding to an element of the lift on the line AD. Hence the sum of $Q_1R + Q_2R_1 + Q_3R_2 +$ . . . $+ DR_m$ will be the total length of the curve corresponding to the total lift BD.

Now let the length of the curve $= l$, the amount of lift $BD = h$, the height AB of the starting-point above the centre of the generating circle $= k$, and the radius CA of this circle $= r$.

Let each element of the lift BQ_1, Q_1Q_2, &c. $= x$, and let $h = mx$.

From equation (3) $Q_1R = \frac{k}{r}x.$

Similarly $$Q_2R_1 = \frac{k+x}{r}x$$

$$Q_3R_2 = \frac{k+2x}{r}x \text{ and so on.}$$

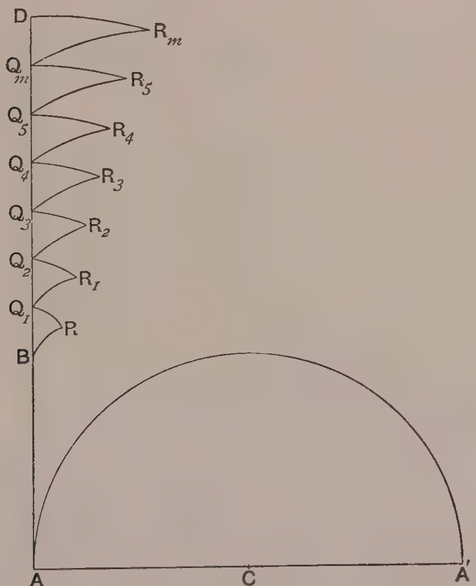


Fig. 133.

Since $BD = h = mx$, the length of curve l corresponding to the lift BD

$$= \left\{k+(k+x)+(k+2x)+ \ . \ . \ . \ +(k+(m-1)x\right\}\frac{x}{r}$$

$$= \left\{mk+x+2x \ . \ . \ . \ +(m-1)x\right\}\frac{x}{r}$$

$$= \left(mk + \frac{m(m-1)x}{2}\right)\frac{x}{r} = \frac{mx}{r}\left(k + \left(\frac{m-1}{2}\right)x\right)$$

$$= \frac{mx}{r}\left(k + \frac{mx}{2} - \frac{x}{2}\right)$$

But mx is equal to h, and when the number of parts is infinitely great, x, becoming infinitely small, disappears.

$$\text{Therefore } l = \frac{h}{r}\left(k + \frac{h}{2}\right)$$

$$= \frac{2kh + h^2}{2r} \qquad (4)$$

The practical importance of this equation in mill designing has already been pointed out. The above expression being a minimum for a given height of lift $\left(\frac{h^2}{2r}\right)$ when $k = O$ and increasing as k increases, it follows that a larger surface of the cam curve is required for equal amounts of lift when the tappet is high up above the centre of the cam shaft than when it is nearer to it. As a concrete example, taking the cam curve drawn on page 206, Fig. 41, the length of curve involved in raising the tappet through the first inch of its lift is

$$\frac{(2 \times 4{\cdot}5 \times 1) + 1}{2 \times 4{\cdot}375} = \frac{10}{8{\cdot}75} = 1{\cdot}14 \text{ inches,}$$

whilst the length corresponding to the final inch of its lift is

$$\frac{\{2 \times (6 + 4{\cdot}5) \times 1\} + 1}{2 \times 4{\cdot}375} = \frac{22}{8{\cdot}75} = 2{\cdot}51 \text{ inches.}$$

In this particular case, therefore, the length of curve involved and the consequent friction generated and power lost during the last inch of lift is 2·2 times as great as during the first inch. Many mill designers are in the habit of constructing a cam arranged to give a lift several inches greater than the maximum amount ever needed in practice, and consider that it is a simple matter to shorten the length of drop when desired, by fixing the tappet further up the stem. It is now obvious how very much power is wasted by such an arrangement; it is the mill man's duty to know what length of drop he will require for milling a given quartz, and to have his cams designed to give exactly that drop and no more, if he wants a machine that shall not waste power. Where different kinds of quartz have to be treated, the cam should be arranged to give the average amount of drop usually required,

and not the maximum drop ever wanted. It is better to crush an occasional parcel rather more slowly than to be working continuously during most of the time under conditions which are mechanically wrong, and involve loss of power. These remarks apply of course with especial force to mills that have not an abundant supply of free water-power.

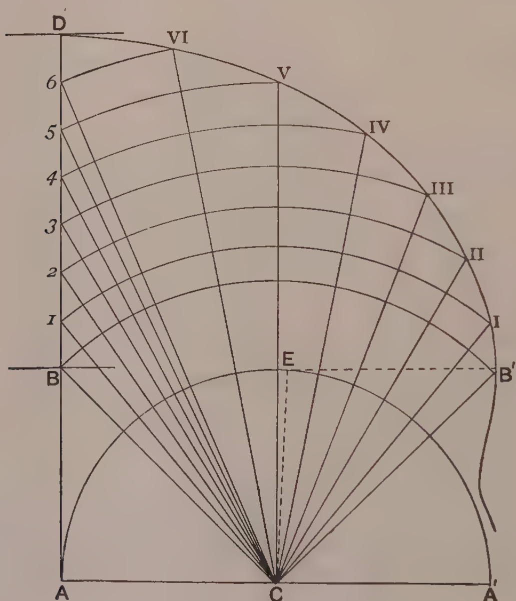
It may be useful to give an alternative method of drawing the involute curve, and one will now be given which depends on that definition of the curve with which this section commences.

The method given previously (page 203) is a geometrical one for setting out the involute, considering this curve as formed by the tangential unwinding of the circumference of a circle. The old method, well known to mill men, which was based on this same property of the curve, was by actually unwrapping a piece of string which had been wound round a thin disc of wood, the radius of which was equal to the distance between the cam shaft and stamp stem axes ; a pencil was inserted in a loop at the end of this string, which was always kept taut. This rough method is not accurate enough for so important a matter, and the curve should always be set out geometrically.

The alternative method is shown in Fig. 134, which is drawn to a scale of three inches to the foot, and is for the same cam as is represented in Fig. 41 (page 206).

Let C be the centre of the cam shaft and AD the centre line of the stamp stems, CA being taken equal to $r(=4{\cdot}375'')$. On AD set off, as before, AB equal to the radius of the cam boss $k(=4{\cdot}5'')$, and BD equal to the lift $h(=7'')$; divide BD into the spaces $B1$, 12, 23 . . . $6D$, each equal to one inch. D is evidently the highest point of the cam curve when the tappet is at the top of its lift. When the tappet was an inch below the top, a corresponding point of the cam curve was at 6 ; we know that during the time that the tappet rose one inch, this point has traversed an angle equal to $\frac{180^\circ}{\pi r} = 13^\circ{\cdot}6'$ as before. Hence the line $C6$ will have travelled through that arc by the time the cam is in its highest position, and the point now corresponding to 6 is foun by joining $C6$, describing a circle with centre C and radius $C6$, and

setting out the angle $6CVI$ equal to $13^\circ{\cdot}6'$, cutting the circle $6VI$ in VI; VI is accordingly a point in the curve. Similarly circles are described with radii $C5$, $C4$ . . . CB, and angles $5CV$, $4CIV$, . . . BCB' are drawn equal respectively to $2(13^\circ{\cdot}6') = 26^\circ{\cdot}12$, $3(13^\circ{\cdot}6) = 39^\circ{\cdot}18'$, . . . $7(13^\circ{\cdot}6') = 91^\circ{\cdot}42' = \dfrac{h180^\circ}{\pi r}$. By



Scale $\frac{1}{3}''$ = 1 ft.

FIG. 134.

then joining the points D, VI, V, IV . . . B', thus found, the cam curve will be obtained, this being a curve such that for each angle of $\dfrac{180^\circ}{\pi r}$ $(= 13^\circ{\cdot}6')$ of revolution round the centre C, it will lift a point one inch on the line AD, the distance CA being equal to r.

It can readily be proved that this curve is the normal involute

of the circle whose centre is at C and radius CA, for in comparing this involute curve as described in Fig. 41 with Fig. 134, let the locus be considered of any similar point in the two curves, such as VII, Fig. 41, and B′, Fig. 134. In Fig. 134 describe the circle AEA' with centre C and radius CA; draw $B'E$ tangent to this circle in E, and join CE. The triangles ABC in both figures are equal by construction.

The line $CB' = CB = CVII$.

The angle $ACB' = ACB + BCB' = ACB + \frac{h180^\circ}{\pi r}$,

but the angle $ACVII = AC7 + 7CAII = AC7 + ACB = \frac{h180^\circ}{\pi r} + ACB$.

Therefore the angle $ACB' = ACVII$, and the line $CB' = CVII$; therefore the loci of the points B' and VII coincide. Similarly it may be proved that any corresponding points of the curves described in Figs. 41 and 134 coincide, and the curves are therefore identical. But the curve in Fig. 41 is the normal involute to the circle drawn with centre C and radius CA, whilst that drawn in Fig. 134 is a curve of uniform lift for uniform angular motion; therefore this latter curve is the normal involute.

In practice it is advisable to set out the curve in full size by both methods, so that any error in the work will be at once detected by the want of agreement between the curves. It need hardly be said that the curve obtained in the second method can be modified precisely in the same way as that obtained by the previous one, whilst it will also be well to assume for the angle corresponding to an inch of lift a value very slightly less than that given by the expression $\frac{180^\circ}{\pi r}$.

APPENDIX B

SPECIFICATIONS FOR TWENTY STAMP GOLD MILL

Crusher, &c.—One Fraser and Chalmers's Improved Blake Pattern Stone Breaker, with jaw plates, heavy cast frame, fly wheels, driving pulley, extended shaft, and all fixtures and fittings. The jaw plates to be of best steel. The jaw opening to be 15 inches by 9 inches.

One Grizzley, or Ore Screen, 4 feet wide by 10 feet long, with rods, nuts, spacing washers, &c., complete. The 10 feet bars to be of taper section, giving an opening at the top of $1\frac{1}{2}$ inches.

Four Fraser and Chalmers's Automatic Ore feeders, of frame pattern, with sheet iron hoppers, revolving feed plates, springs, brake wheels, wings, and all accessories. Feed plate liners of mild steel to be supplied of removable pattern.

Four Steel Feed Collars, bored and fitted to stems with set screws, for actuating the feeders.

Stamps.—Twenty Stamp Battery of 1,050 lbs. weight each stamp, arranged to be driven by four cam shafts, by belts and tighteners from stamp countershaft. The Battery being complete in detail as follows:—

Four Fraser and Chalmers's latest pattern Homestake Mortars, of hard close-grained cast iron, planed upon bottom with foundation bolt holes drilled to template, dressed for screen frames, and fitted for copper lining. Mortar to be complete with screen, of round or slot punched Russia iron or wire cloth as may be desired, screen frame of hard wood, screen frame keys of wrought iron,

chuck blocks of wood with copper lining fitted and fixed, chuck block keys of wrought iron, steel liners, foundation bolts, back copper liner, &c., &c.

Four pieces of best quality Rubber Packing for mortar bottom, for mortar to rest upon, punched for foundation bolts to correspond with mortar bottom.

Twenty Refined Iron Stamp Stems, having both ends tapered so as to be reversible. The stems to be turned full length and fitted to heads.

Twenty Best Crucible Cast Steel Tappets, bored for stems, and fitted to suit the cams, and to be reversible, with gibs and keys. Tappets to have hard, broad faces.

Twenty Best Crucible Cast Steel Stamp Heads, with drift holes for stems and shoes, recessed for the head of the shoe and for the tapered ends of the stems.

Twenty Hammered Forged Steel Battery Shoes of Fraser and Chalmers's special mixture of steel.

The weight of the Stamp is made up as follows :—

Stem	448 lbs.
Tappet	155 ,,
Head	234 ,,
Shoe	213 ,,
Total	1,050 lbs.

Twenty Hammered Forged Steel Battery Dies of Fraser and Chalmers's special mixture of steel.

Four Refined Iron or Steel Cam Shafts, with keyway cut for cam shaft pulleys, each with two collars and set screws; the cam shafts being turned full length and drilled for cams.

Four Cam Shaft Pulleys, built up of wood segments on a pair of cast iron sleeve flanges keyed to the cam shafts, the flanges being bolted through the wood before the pulley is turned up to its requisite diameter, with crown face, and then painted.

Four Outside Cam Shaft Boxes, of close hard cast iron, 12 inches long, fitted with babbitt metal, well hammered to make it fit perfectly into the casting, and then bored for cam shaft. The

bearings to be complete with caps and bolts for same, and to be faced for cam shaft collars.

Two Double Middle Cam Shaft Boxes, of close hard cast iron, 20 inches long, fitted with babbitt metal, well hammered to make it fit perfectly into the casting, and then bored for cam shaft. The bearings to be complete with caps and bolts for same, and to be faced for cam shaft collars.

Twenty Best Crucible Cast Steel Cams of double arm pattern, accurately bored and fitted to cam shafts, with Blanton patent wedges.

Four Jack Shafts of rolled iron for hanging up the stamps.

Eight Jack Shaft Chairs or bearings of cast iron, with lag screws for fixing to battery timbers.

Twenty Sockets for finger pieces, of cast iron, leather lined.

Twenty Wooden Finger Pieces for holding up stamps, dressed to fit the sockets, and each fitted with wrought iron tips and malleable iron handles.

Four Sets of Hard Wood Guides, bored for stems, with bolts, nuts and washers complete. Each set comprising the two sections for top and bottom guides of five stamps.

One Complete Set of Wrought Iron, Lap-Welded, Screwed and Coupled, Water Pipes for 20 stamp battery, with all valves, tees, reducers, bends and connections for feeding to mortars, and main supply pipe of a length to connect to a tank supposed to be immediately outside the mill building, the main being sufficently large for the full supply of a 20 stamp mill.

Four Lengths of Hose for washing the amalgamated copper plates, with bib connections to the water service of the battery.

Coppers.—Four Outside Copper Plates 12 feet long by width of mortar, by $\frac{1}{8}$ inch thick, to be of best soft rolled smooth surface copper for amalgamating purposes, to be arranged before each 5 stamps without resting upon or against the mortars, so as to be as free as possible from jar.

Crawl.—One Overhead Carriage Crawl and Track Iron for the same, with wood screws for fixing to crawl beams.

One set of Differential Pulley blocks of 30 cwt. capacity, with chain tackle for the same, for handling the stamp parts.

AMALGAM SAFE, &c.—One Amalgam Safe, with sheet iron sides, with strainer and padlock.

RETORT, &c.—Ten inch Gold Retort, dressed out inside, and properly fitted with cover, bale, wedge and condenser pipe.

One Complete Set of Ironwork for a bullion furnace 16 inches diameter, including bars, doors, &c.

CLEAN-UP PAN—One Fraser and Chalmers's Clean-up Pan of 30 inches diameter, with iron sides, and with step pulley for driving the same at variable speeds.

TRANSMISSION, &c.—One complete Set of Countergear for the whole of the above machinery, in accordance with drawings to be supplied by us, including:—

One Main Line Shaft, with receiving pulley or half coupling to correspond with power connection.

One Clean-up Pan Countershaft.

All the above to be complete with pulleys, keyways and keys, bearings, couplings, collars, set screws, belting and lace leather for driving the whole of the above machinery from steam or water power as specified.

Four Belt Tighteners for battery belts, with shafts, hand wheels, racks, pinions, &c.

One Belt Tightener, with shaft, hand wheel, rack, pinion, &c., for crusher belt.

BUILDING BOLTS.—One Complete Set of Nuts, Bolts, Rods and Washers, Straps, Angles, &c., &c., forming all the ironwork for the buildings to enclose this machinery; the buildings supposed to be of lumber.

Total weight, approximately, 57 tons 3 cwts.

Price £ .

FRASER AND CHALMERS, LTD.,
London.

SPECIFICATIONS FOR BATTERY FRAMEWORK

One Complete Set of Timbers for Twenty Stamps Framework, of A design or pattern, cut from best selected Georgia Pitch Pine, specially imported for Battery Framework use ; comprising Mortar Foundations, Mortar Block Binders, Battery Posts, Battery Post Sills, Mudsills, Guide Beams, Back Struts, Feed Floor Posts, Joists and Supports, Tightener Guides and Supports, &c., all properly cut to size, dressed, mortised, tenoned and fitted, Bolt Holes drilled, erected and marked to place, after which the framework to be properly painted, taken down and protected for shipment.

One Complete Set of Bolts, Rods, Turnbuckles, Nuts and Washers, for Battery Framework complete, including Mortar Foundation Bolts.

Total weight, approximately, 33 tons.

Price £ .

FRASER AND CHALMERS, LTD.,
London.

TABLE I.—*Showing the theoretical horse-power developed by given quantities of water at given heights of fall.*

Cubic feet per minute.	Height of Fall in Feet.											
	5	10	20	30	40	50	60	70	80	90	100	200
10	0·095	0·189	0·378	0·568	0·757	0·946	1·135	1·324	1·513	1·703	1·892	3·783
20	0·189	0·378	0·757	1·135	1·513	1·892	2·270	2·648	3·027	3·405	3·783	7·567
30	0·284	0·568	1·135	1·703	2·270	2·838	3·405	3·973	4·540	5·108	5·675	11·350
40	0·378	0·757	1·513	2·270	3·027	3·783	4·540	5·297	6·053	6·810	7·567	15·133
50	0·473	0·946	1·892	2·838	3·783	4·729	5·675	6·621	7·567	8·513	9·458	18·917
100	0·946	1·892	3·783	5·675	7·567	9·458	11·350	13·242	15·133	17·025	18·917	37·833
200	1·892	3·783	7·567	11·350	15·133	18·917	22·700	26·483	30·267	34·050	37·834	75·667
300	2·837	5·675	11·350	17·025	22·700	28·375	34·050	39·725	45·400	51·075	56·750	113·500
400	3·783	7·567	15·133	22·700	30·266	37·833	45·400	52·966	60·533	68·099	75·667	151·333
500	4·729	9·458	18·917	28·375	37·833	47·292	56·750	66·208	75·667	85·125	94·583	189·167
600	5·675	11·350	22·700	34·050	45·400	56·750	68·100	79·450	90·800	102·150	113·500	227·000
700	6·621	13·242	26·483	39·725	52·966	66·208	79·450	92·691	105·933	119·175	132·416	264·833
800	7·566	15·133	30·267	45·400	60·533	75·667	90·800	105·933	121·067	136·200	151·333	302·667
900	8·509	17·024	34·049	51·075	68·099	85·125	102·150	119·174	136·200	153·225	170·249	340·500
1000	9·458	18·916	37·833	56·750	75·667	94·583	113·500	132·416	151·333	170·250	189·166	378·333
2000	18·916	37·833	75·667	113·500	151·333	189·167	227·001	264·834	302·667	340·501	378·334	756·667

TABLE II.—*Showing the yield of bullion per ton of ore corresponding to the weights of prills obtained from an assay on 500 grains of ore.*

Weight of prills in grains.	Yield of Bullion.					
	Per Ton of 2,240 lbs.			Per Ton of 2,000 lbs.		
	oz.	dwt.	grs.	oz.	dwt.	grs.
0·00025	0	0	7·8	0	0	7
0·0005	0	0	15·7	0	0	14
0·001	0	1	7·4	0	1	4
0·002	0	2	14·7	0	2	8
0·003	0	3	22·1	0	3	12
0·004	0	5	5·4	0	4	16
0·005	0	6	12·8	0	5	20
0·006	0	7	20·2	0	7	0
0·007	0	9	3·5	0	8	4
0·008	0	10	10·9	0	9	8
0·009	0	11	18·2	0	10	12
0·010	0	13	1·6	0	11	16
0·020	1	6	3·2	1	3	8
0·030	1	19	4·8	1	15	0
0·040	2	12	6·4	2	6	16
0·050	3	5	8·0	2	18	8
0·060	3	18	9·6	3	10	0
0·070	4	11	11·2	4	1	16
0·080	5	4	12·8	4	13	8
0·090	5	17	14·4	5	5	0
0·100	6	10	16	5	16	16
0·200	13	1	8	11	13	8
0·300	19	12	0	17	10	0
0·400	26	2	16	23	6	16
0·500	32	13	8	29	3	8
0·600	39	4	0	35	0	0
0·700	45	14	16	40	16	16
0·800	52	5	8	46	13	8
0·900	58	16	0	52	10	0
1·000	65	6	16	58	6	16
2·000	130	13	8	116	13	8
3·000	196	0	0	175	0	0

TABLE III.—*Showing the value of gold per ounce according to its fineness.*

Fineness in parts per 1000.	Value per Ounce.							
	Sterling.						American Currency.	
	Bank Rate.			Mint Rate.				
	£	s.	d.	£	s.	d.	$	cts.
1000	4	4	9·82	4	4	11·45	20	67·21
900	3	16	4·04	3	16	5·50	18	60·49
800	3	7	10·26	3	7	11·56	16	53·77
700	2	19	4·47	2	19	5·62	14	47·05
600	2	10	10·69	2	10	11·67	12	40·33
500	2	2	4·91	2	2	5·73	10	33·61
400	1	13	11·13	1	13	11·78	8	26·88
300	1	5	5·35	1	5	5·84	6	20·16
200	0	16	11·56	0	16	11·89	4	13·44
100	0	8	5·78	0	8	5·95	2	06·72
90	0	7	7·60	0	7	7·75	1	86·05
80	0	6	9·43	0	6	9·56	1	65·38
70	0	5	11·25	0	5	11·36	1	44·70
60	0	5	1·07	0	5	1·17	1	24·03
50	0	4	2·89	0	4	2·93	1	03·36
40	0	3	4·71	0	3	4·78	0	82·69
30	0	2	6·54	0	2	6·58	0	62·02
20	0	1	8·36	0	1	8·39	0	41·34
10	0	0	10·18	0	0	10·19	0	20·67
9	0	0	9·16	0	0	9·18	0	18·60
8	0	0	8·14	0	0	8·16	0	16·54
7	0	0	7·13	0	0	7·14	0	14·47
6	0	0	6·11	0	0	6·12	0	12·40
5	0	0	5·09	0	0	5·10	0	10·34
4	0	0	4·07	0	0	4·08	0	8·27
3	0	0	3·05	0	0	3·06	0	6·20
2	0	0	2·04	0	0	2·04	0	4·13
1	0	0	1·02	0	0	1·02	0	2·07
0·9	0	0	0·92	0	0	0·92	0	1·86
0·8	0	0	0·81	0	0	0·82	0	1·65
0·7	0	0	0·71	0	0	0·71	0	1·45
0·6	0	0	0·61	0	0	0·61	0	1·24
0·5	0	0	0·51	0	0	0·51	0	1·03
0·4	0	0	0·41	0	0	0·41	0	0·83
0·3	0	0	0·31	0	0	0·31	0	0·62
0·2	0	0	0·20	0	0	0·20	0	0·41
0·1	0	0	0·10	0	0	0·10	0	0·21

TABLE IV. *Conversion of grammes per metric ton into ounces, &c. per long and short ton.*

Grammes per metric ton.	Per short ton (2,000 lbs.).			Per long ton (2,240 lbs.).		
	oz.	dwts.	grs.	oz.	dwts.	grs
1,000	29	3	8	32	13	8
900	26	5	0	29	8	0
800	23	6	16	26	2	16
700	20	8	8	22	17	8
600	17	10	0	19	12	0
500	14	11	16	16	6	16
400	11	13	8	13	1	8
300	8	15	0	9	16	0
200	5	16	16	6	10	16
100	2	18	8	3	5	8
90	2	12	12	2	18	19
80	2	6	16	2	12	7
70	2	0	20	2	5	18
60	1	15	0	1	19	5
50	1	9	4	1	12	16
40	1	3	8	1	6	3
30	0	17	12	0	19	14
20	0	11	16	0	13	2
10	0	5	20	0	6	13
9	0	5	6	0	5	21
8	0	4	16	0	5	5
7	0	4	2	0	4	14
6	0	3	12	0	3	22
5	0	2	22	0	3	6
4	0	2	8	0	2	15
3	0	1	18	0	1	23
2	0	1	4	0	1	7
1	0	0	14	0	0	15·7
0·9	0	0	12·6	0	0	14·1
0·8	0	0	11·2	0	0	12·5
0·7	0	0	9·8	0	0	10·9
0·6	0	0	8·4	0	0	9·4
0·5	0	0	7·0	0	0	7·8
0·4	0	0	5·6	0	0	6·2
0·3	0	0	4·2	0	0	4·7
0·2	0	0	2·8	0	0	3·1
0·1	0	0	1·4	0	0	1·6

TABLE V.—*Conversion of ounces, &c., per long and short ton into grammes per metric ton.*

Ounces, &c., per ton.	Grammes per metric ton. A.	Grammes per metric ton. B.	Ounces, &c., per ton.	Grammes per metric ton. A.	Grammes per metric ton. B.
ozs. dwts. grs.			oz. dwts. grs.		
100 0 0	3,428·57	3,061·22	0 7 0	12·00	10·71
90 0 0	3,085·71	2,755·10	0 6 0	10·29	9·18
80 0 0	2,742·86	2,448·98	0 5 0	8·57	7·65
70 0 0	2,400·00	2,142·86	0 4 0	6·86	6·12
60 0 0	2,057·14	1,836·73	0 3 0	5·14	4·59
50 0 0	1,714·29	1,530·61	0 2 0	3·43	3·06
40 0 0	1,371·43	1,224·49	0 1 0	1·71	1·53
30 0 0	1,028·57	918·37	0 0 23	1·64	1·47
20 0 0	685·71	612·24	0 0 22	1·57	1·40
10 0 0	342·86	306·12	0 0 21	1·50	1·34
9 0 0	308·57	275·51	0 0 20	1·43	1·28
8 0 0	274·29	244·90	0 0 19	1·36	1·22
7 0 0	240·00	214·29	0 0 18	1·29	1·16
6 0 0	205·71	183·67	0 0 17	1·22	1·08
5 0 0	171·43	153·06	0 0 16	1·14	1·02
4 0 0	137·14	122·45	0 0 15	1·07	0·96
3 0 0	102·86	91·84	0 0 14	1·00	0·89
2 0 0	68·57	61·22	0 0 13	0·93	0·83
1 0 0	34·29	30·61	0 0 12	0·86	0·77
0 19 0	32·57	29·08	0 0 11	0·79	0·70
0 18 0	30·86	27·55	0 0 10	0·71	0·64
0 17 0	29·14	26·02	0 0 9	0·64	0·58
0 16 0	27·43	24·49	0 0 8	0·57	0·51
0 15 0	25·71	22·96	0 0 7	0·50	0·44
0 14 0	24·00	21·43	0 0 6	0·43	0·38
0 13 0	22·29	19·90	0 0 5	0·36	0·32
0 12 0	20·57	18·37	0 0 4	0·28	0·25
0 11 0	18·86	16·84	0 0 3	0·21	0·19
0 10 0	17·14	15·31	0 0 2	0·14	0·13
0 9 0	15·43	13·28	0 0 1	0·07	0·065
0 8 0	13·71	12·24			

NOTE.—If the ounces, &c., are given on the short ton, the corresponding number of grammes will be found in column A; if on the long ton, in column B.

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INDEX

THE END.

RICHARD CLAY AND SONS, LIMITED, LONDON AND BUNGAY.